(Frontispiece.)

L. D. Ricketts.

TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING ENGINEERS

VOL. LV

CONTAINING PAPERS AND DISCUSSIONS OF THE ARIZONA MEETING, SEPTEMBER, 1916

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PREFACE

In this volume most of the papers have to do with the mineral industry of the State of Arizona and it has, therefore, been deemed wise by the Board of Directors to call it the "Arizona Volume." An additional reason for using this name is that all the papers and discussions therein were presented at the Arizona Meeting in September, 1916. So many papers were presented at this meeting that it was impossible to include them all in one volume and a few, therefore, not closely related to the mineral industry of Arizona itself, have been reserved for Volume LVI. The papers in question deal especially with iron and steel, petroleum and gas, and related subjects. The present volume, which is the second published in 1917, contains all the papers of the Arizona Meeting relating to the geology, mining, milling and smelting practice of that State, together with papers on non-ferrous mining and metallurgical subjects, some describing the practice in other districts and other papers treating, in a general way, of some process or method used in, or applicable to, the mining and treatment of ores of the Southwest.

At the Arizona Meeting, for the first time in the history of the Institute, a full session was devoted to the subject of Flotation. The carefully prepared papers on this timely subject and the interesting and valuable discussion thereof worthily mark this important epoch in metallurgical history.

For the excellent series of papers on Arizona practice we are indebted chiefly to L. D. Ricketts, President of the Institute at the time of the Arizona Meeting, whose energetic solicitation was ably seconded by the Executive Committee of the Arizona Local Section. It is a pleasure to record our appreciation of the splendid work of Dr. Ricketts and his friends of the Southwest in enriching the literature of the Institute.

Volumes LV and LVI will contain all the papers printed in the monthly Bulletin during the year beginning March, 1916, but no entire Bulletin is superseded by this one volume.

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PAPERS

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PROCEEDINGS OF THE ONE-HUNDRED AND THIRTEENTH MEETING, ARIZONA

Sept. 17 *to* 25 *inclusive,* 1916

GENERAL COMMITTEE

GERALD F. G. SHERMAN, *Chairman.*
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Committee at El Paso

W. D. GORDON, ROBERT McCART, JR., W W. ROSE

Committee at Santa Rita

JOHN M. SULLY, *Chairman.*

Committees at Douglas

Reception	*Train Movement*	*Technical Session*
H. A. CLARK,	J. O. AMBLER,	H. O. HAMMOND,
G H. DOWELL,	ROBERT RAE	FOREST RUTHERFORD.
A V. DYE		

Committee in the Globe-Miami Section

Local Committee

L. O. HOWARD, Globe, F W. MACLENNAN, Miami, *Joint Chairmen*

Entertainment of Ladies

MRS. B. B. GOTTSBERGER,
MRS I. N. BARKDOLL,
MRS. L O HOWARD,
MRS. W. B CRAMER

Technical Sessions and Banquet

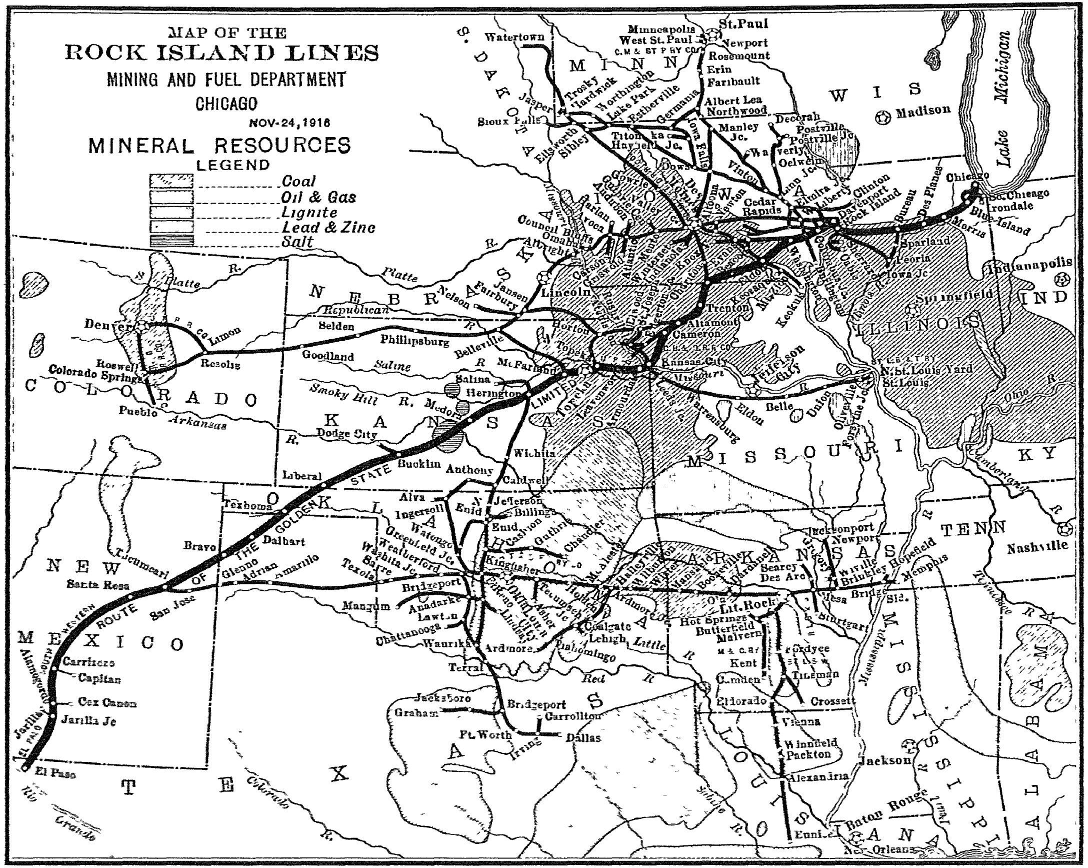
J. H. HENSLEY, JR.
G N BJORGE

Transportation, Automobiles, Etc.

I N. BARKDOLL, H. P. BOWEN, W. F. GEIGER

Bureau of Information

ROBERT FRANKE.


MAP OF THE
ROCK ISLAND LINES
MINING AND FUEL DEPARTMENT
CHICAGO
NOV-24,1916
MINERAL RESOURCES
LEGEND
Coal
Oil & Gas
Lignite
Lead & Zinc
Salt
Chicago
Lake Michigan
Madison
Minneapolis
St.Paul
Des Moines
Omaha
Lincoln
Denver
Colorado Springs
Topeka
Kansas City
St.Louis
Indianapolis
Springfield
Wichita
Dodge City
Liberal
Dalhart
Amarillo
Tucumcari
Santa Rosa
El Paso
Little Rock
Memphis
Nashville
Ft. Worth
Dallas
Alexandria
New Orleans
Jackson
Baton Rouge
THE GOLDEN STATE LIMITED
ROUTE OF THE GOLDEN STATE LIMITED
NEW MEXICO
COLORADO
NEBRASKA
KANSAS
OKLAHOMA
TEXAS
MISSOURI
ILLINOIS
IOWA
MINN
WIS
S. DAKOTA
ARKANSAS
LOUISIANA
MISSISSIPPI
ALABAMA
TENN
KY
IND

Such extraordinary developments in mining and metallurgy and so many new departures have been made in mining, concentration and smelting in the State of Arizona in recent years that extraordinary means had to be taken to cover the more important points of interest in the State. A special train to carry the party from point to point during the hours of darkness, so that daylight might be used to the utmost advantage, was therefore provided. Even with this advantage, it was impossible to visit all the camps, and it was with great regret that the main party was obliged to omit such interesting places as the United Verde Copper Co. and the United Verde Extension Mining Co. at Jerome, the Ray Consolidated Copper Co. at Ray, the Clifton-Morenci District, and the new developments of the Gold Road-Oatman District.

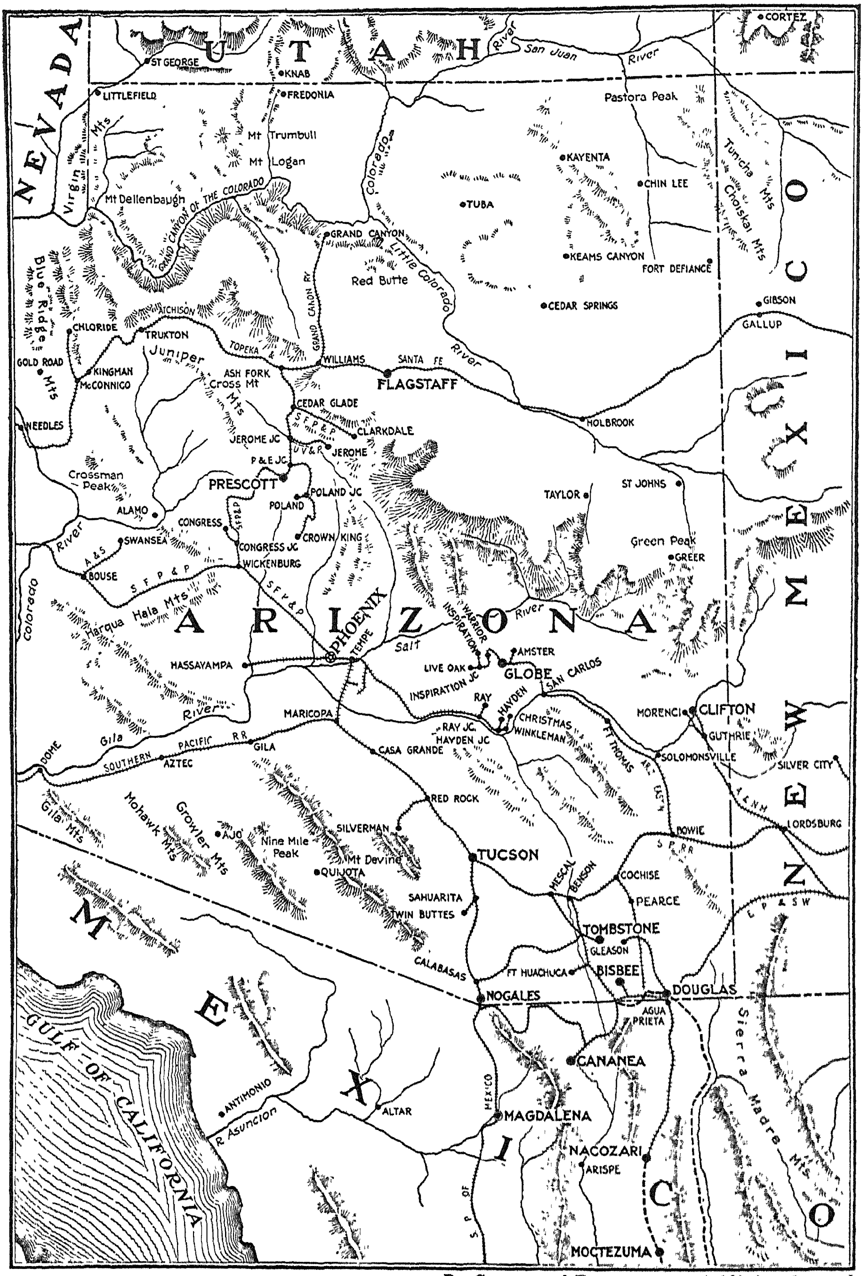
Two special cars started from New York City carrying members of the party as far as Chicago. There the numbers were augmented by members from different points. The Chairman and representatives of the Chicago Local Section met the party and accompanied it on an interesting drive through Washington and Jackson Parks in automobiles. A banquet of about 75 persons at the La Salle Hotel followed.

Three special Pullman cars, the private car "Anaconda" belonging to Past-President Benjamin B. Thayer, and a special combination baggage and club car for the accommodation of the Institute party left Chicago on the Golden State Limited of the Chicago, Rock Island and Pacific Railroad, but some time before morning was made up as the Institute Special which then traveled as a separate unit through to the Grand Canyon, Ariz. Through the kindness of Mr. Carl Scholz, large geological maps showing the mineral resources of the Rock Island Lines between Chicago and El Paso were distributed to all members of the party and added greatly to the interest of the trip. (See page viii.)

Additional members were picked up at Kansas City and other points and the special train arrived in El Paso at 2:40 p. m. on Sunday, Sept. 17. At this point parties from Montana, Utah, Colorado, California, and other places, joined the gathering, as well as more than 25 members from Arizona, including the Arizona Committee. Several El Paso members, under the leadership of their very efficient Committee, had arranged for a visit by automobiles to the National Guard camps—containing, it was said, about 50,000 militia from different States—to Fort Bliss, and to the El Paso Smelter of the American Smelting and Refining Co. Here the party had an opportunity to see a lead smelter which had been transformed to a copper smelter to meet commercial conditions and also a Peirce-Smith copper converter 13 ft. in diameter. Some of the members motored over to Old Mexico and collected souvenirs and saw a bull fight.

After a brief period for rest the members and guests were tendered by the El Paso members a most interesting Mexican supper at the Toltec Club, at which were present 179 persons and which was enlivened by the music of a Mexican band. The menu of this supper follows: Toronja; Enchiladas con Huevos; Tamales de Pollo; Tortillas; Frijoles Refritos; Nieve Napolitana; Quequi; Cafe. To each menu card was pinned a small Mexico sombrero made of horse hair.

The party leaving El Paso on Sunday evening numbered 150 persons, and the arrangements from that point were in the capable hands of the Arizona General Committee. Attractive, well printed, and well illus-



By Courtesy of Engineering and Mining Journal.

MAP SHOWING POINTS IN ARIZONA VISITED DURING THE SEPTEMBER, 1916, MEETING. (Santa Rita and Hurley, N. M., where are located the mine and mill of the Chino Copper Co., were also visited, but are not shown on the map.)

trated general programs of the meeting were distributed which contained a map of Arizona showing the principal points of mining and metallurgical interest. In this booklet there were also some statistics and information regarding the mines of the Chino Copper Co. and of the Warren District and the smelters at Douglas. Throughout the entire trip the admirable arrangements of the General Committee and the Local Committees of the different districts were a frequent source of comment. Not only was every possible comfort and convenience provided, but carefully prepared data on each of the districts were given to the visitors upon arrival.

Monday morning, Sept. 18, found the party at Santa Rita, N. Mex., where the members were awakened by a miners' salute consisting of a

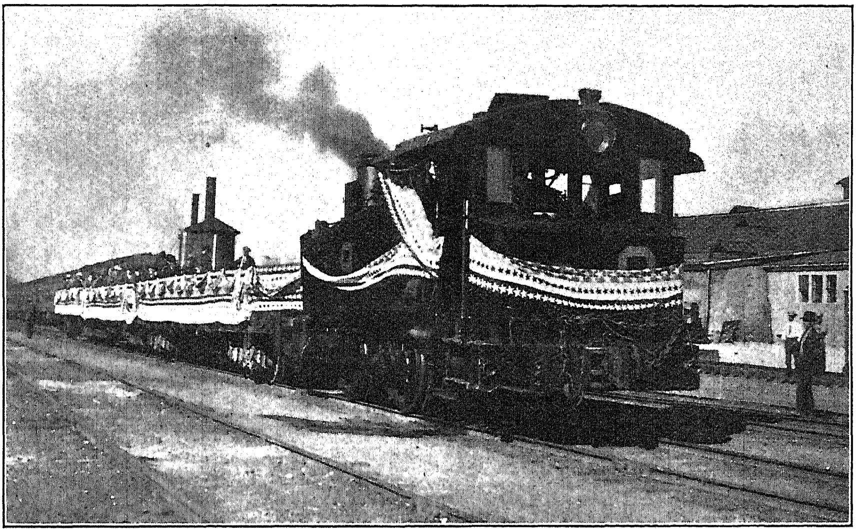


FIG. 1.—THE OBSERVATION TRAIN WHICH CARRIED THE INSTITUTE PARTY AROUND THE CHINO COPPER CO. OPEN-CUT WORKINGS.

series of blasts at the mine of the Chino Copper Co. Upon detraining the party boarded an observation train elaborately decorated for the occasion, as shown in Fig. 1, and was taken around the open-cut workings of the Chino mine, estimated to contain about 90 million tons of porphyry ore averaging about 1.75 per cent. copper. This property is said to be the first in the Southwest to be mined by steam shovel, and, in the year 1915, produced an average of over 200,000 lb. of copper per day.

What can be described only as a fleet of automobiles, which had been gathered from mining districts within a radius of 30 miles, was then ready to take the members to Hanover, where are located the mine and mill of the Empire Zinc Co. These arrangements are typical of the thoroughness and thoughtfulness with which the Committees at all the different points had arranged to give their visitors the maximum opportunity to see the points of technical interest with the minimum of fatigue. Those who took this trip were especially interested in the Rowand-Wetherill magnetic separator treating an ore con-

taining zinc and lead. Those of the party who did not go to Hanover visited the workings of the Chino mine and also the large preliminary crushing plant at the mine, but all foregathered at Santa Rita about noon to view the "Chino Bar for Ladies and Gentlemen" and the interesting relics of ancient Spanish mining, and to listen to the music provided by the excellent band of the Eleventh U. S. Cavalry. The tennis courts of the Chino Copper Co. had been covered by a large tent decorated with bunting and laid with tables for 200 guests. Here a splendid barbecue was enjoyed by the visitors. The party then had the option of going either by automobile or by the special train to Hurley, where is located the mill of the Chino company, giving the party its first view of flotation work on a large scale, as well as an opportunity to see tailing dams constructed to conserve both the waste water and the tailings

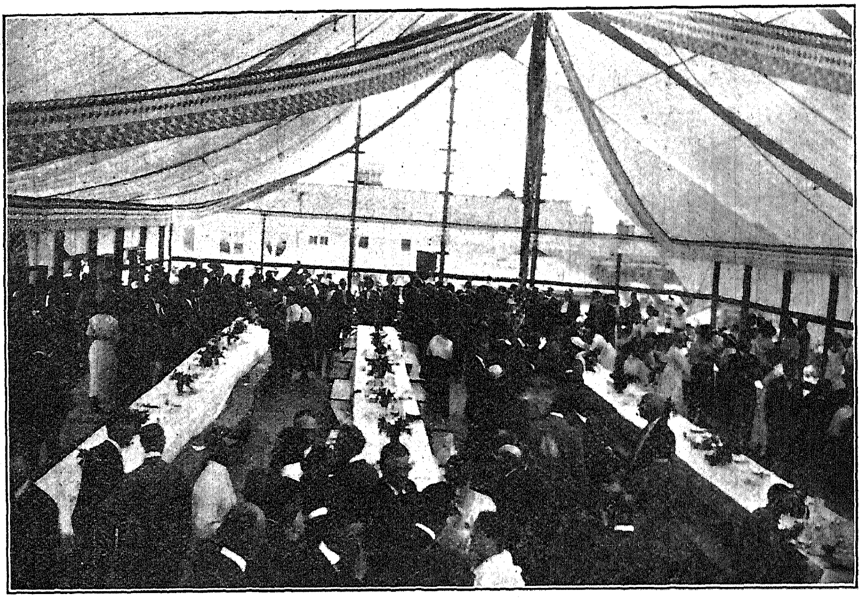


FIG. 2.—SCENE OF THE BARBECUE AT SANTA RITA, N. M.

which are to be subsequently treated in a flotation plant now being constructed.

A splendid dancing pavilion had been erected next to the tennis courts at Hurley, and after dinner many members of the party enjoyed dancing to the music of the Eleventh U. S. Cavalry band. The first day of the meeting closed leaving an impression in the minds of all that it would be very difficult to exceed the interest and pleasure that had been afforded.

Tuesday morning, Sept. 19, the Institute special train of 13 cars arrived at Douglas and was greeted by the Sixth U. S. Artillery band and another fleet of automobiles waiting to take the party to the Copper Queen smeltery. Some members took advantage of this opportunity and others went by the Institute special train, which was carried over the company's tracks and waited at the smeltery until the party had finished its visit there and then carried all the visitors to the plant of the

Calumet and Arizona Co. All members of the party were particularly struck by the cleanliness of these smelters, as evidence of which may be mentioned that a buffet luncheon was served under the dust chambers of the Calumet and Arizona plant. After luncheon the party was carried by train to the Y. M. C. A. building, where a technical session was held on the subject of "Smelting." The ladies were entertained by an automobile ride and tea at the residence of Mrs. Forest Rutherford. In the evening the second technical session, on the subject of "Leaching," was held, and the ladies were entertained with dancing and a band concert at the Country Club. It is interesting to note that, in spite of the other attractions, the attendance at the technical sessions was invariably large, there being 175 persons at the opening session and 150 persons at the evening session.

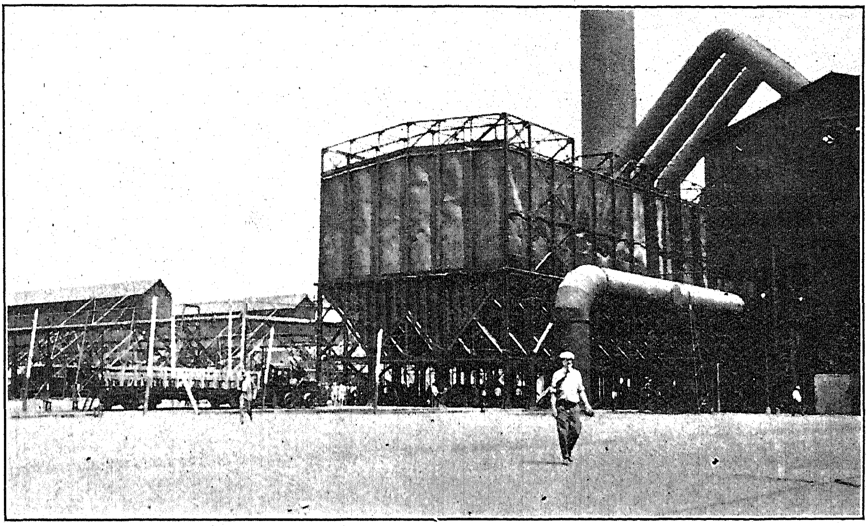


FIG. 3.—SHOWING THE DUST BINS OF THE CALUMET AND ARIZONA COMPANY, DOUGLAS, ARIZ., UNDER WHICH A BUFFET LUNCHEON WAS SERVED.

The 30 miles to the Warren District were made at night and the special train arrived at Lowell in the morning, where the members were given a choice of four different trips, viz.: (1) Underground inspection of the Calumet and Arizona Mining Co.'s Junction shaft and thence to the Briggs mine and the Tintown vein; (2) Neptune tunnel of the Copper Queen Mine; (3) the Sacramento main shaft or (4) a surface trip, including the hoisting plant of the Sacramento shaft, the surface ore-loading plant, the central power house, the central tool-sharpening plant and the central sawmill of the Copper Queen, the Oliver power plant and the Junction power plant of the Calumet and Arizona, together with the underground pumping stations of the latter company, which handles the water from all the mines of the district.

In the afternoon a technical session on "Geology and Mining" was held in the main auditorium of the high school, there being about 300 persons present. The ladies were entertained at luncheon at the Copper Queen Hotel. After a brief period for rest, the members of the party

were taken to the Warren District Country Club. Adjacent to this club is Camp John C. Greenway of the Twenty-second Regiment of U. S. Infantry and the District of Columbia militia. This regiment gave a dress parade in honor of the Institute party, affording a beautiful sight at the mouth of the Canyon overlooking the broad San Pedro Valley as the sun sank behind the distant mountains. This was followed by a banquet at the Country Club, attended by 280 persons. This banquet was a truly memorable occasion and reminded those members who were with the Institute party at the stop in Arizona in the year 1899 of the extraordinary entertainment afforded them on that occasion.* The food had to be brought by automobile from Bisbee, but was served in perfect condition notwithstanding that the number of persons greatly exceeded the expectations of the Local Committee and equally overtaxed the facilities of the Club.

Philip N. Moore, Vice-President of the Institute, acted as toastmaster and the following toasts were responded to by the speakers indicated:

"Welcome to Arizona," C. T. Knapp.
"Arizona, Old and New," L. D. Ricketts.
"Arizona Present," Walter Douglas.
"The American Mining Congress," Carl Scholz.
"Why We Are Here," Colonel Tillson.
"Ave et Vale," John Mason Ross.

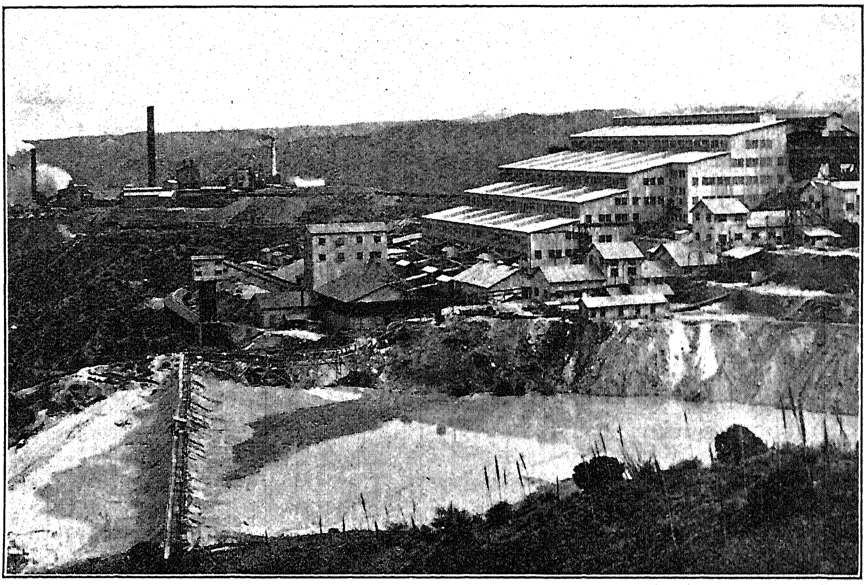


FIG. 4.—THE INSPIRATION CONCENTRATOR WITH THE INTERNATIONAL SMELTING PLANT IN BACKGROUND.

The special train met the party at a near-by junction and to the other 13 cars was now added the private car "Nacozari" of Mr. Walter Douglas. The train proceeded by night to Globe, Ariz. Here, on Thurs-

*Described in *Trans.*, vol. 29, p. lxxxviii.

day morning, Sept. 21, automobiles were in waiting to convey the party to the Old Dominion Copper Mining and Smelting Co's works, where members had an opportunity to see the basic converter which had already been described by L. O. Howard and in which 70,000,000 lb. of copper had been made without relining. In the afternoon a technical session was held at the Martin theater on the subject of "Concentration and Flotation," attended by about 200 persons, while the ladies were entertained at the Cobre Valle Country Club. The session was followed by a meeting of Secretaries of Local Sections, of whom nine were present. The Board of Directors met at dinner at the Old Dominion Hotel, and a technical session on the subject of "Mining" was held later at the Martin Theater, at which 220 persons were present.

On Friday morning, Sept. 22, the special train conveyed some of the party, and others went in automobiles, to the mine of the Inspiration Consolidated Copper Co. and the reduction works of the International

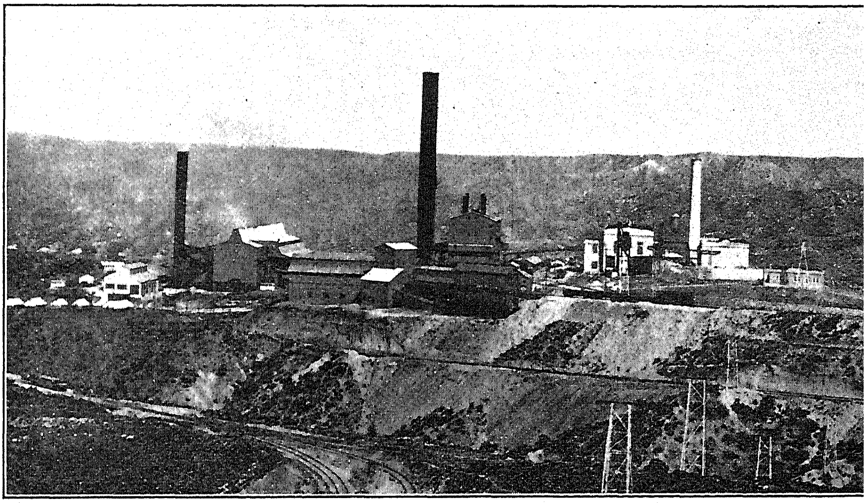


Fig. 5.—Plant of the International Smelting Co., Miami, Ariz.

Smelting Co. This occasion was the greatest technical visit of the meeting. The plants have been so well described already in our *Transactions* that further comment is unnecessary here. To the admirable arrangements of the Local Committee is due the fact that ample opportunity was given to take in the many different points of interest in the very limited time available. An abundant supply of automobiles was available at all times to facilitate the handling of the party, while the special train took care of the main group. After the morning spent at the Inspiration and International plants, the visitors were taken to the mine and mill of the Miami Copper Co., where an excellent buffet luncheon was served at the Club House, followed by a visit to the plants. The company has good reason to be proud of the fine club house provided for its employees, and the Institute party has equally reason to congratulate itself upon having so comfortable a place for luncheon and for the technical session on "Fine Grinding," which occurred at 4:00 p. m. This session was attended by 160 persons who carried on so lively a discussion

that the Chairman was obliged to cut it short in order that the members might take the special train for Globe, which left at 5:30 p. m.

While the men were inspecting the mine and mill and attending the technical session, the ladies were entertained at the bowling alleys of the club house and were then given an opportunity to swim in the crystal-like swimming pool.

At 7:30 p. m. more than 200 members and guests assembled in the auditorium of the Globe high school for the final banquet of the meeting. Sidney J. Jennings, First Vice-President of the Institute, acted as toastmaster and toasts were responded to as follows:

"Safety First," John E. Bacon.
"The Growth of the Globe District," L. D. Ricketts.
"Why the Young Engineer Should Join the Institute," E. P. Mathewson.
"The Utilization of Waste Products," F. G. Cottrell.

The meeting resolved itself into a spontaneous tribute to President Ricketts. His address as reported by the local paper follows:

"While copper and silver in the Globe District may have been known at an earlier date, the earliest actual records show that the Globe and Globe Ledge claims were located as silver mines in 1873. They were not worked, however, and in 1875 some rich silver mines were discovered, the Globe District was organized, and silver ores were milled and silver bullion shipped.

"The first copper was produced at Wheatfields from ore from the Hoosier claim, which was smelted in Mexican adobe furnaces in 1878. The campaign was but a short one, and some 40 tons of black copper pigs was the result. A little later on Mr. John Williams, Sr., worked the Carrie mine and smelter. His ore was a quartzite containing 6 per cent. copper and little or no iron. It is said that he purchased his flux from the big iron outcrop on the Globe claim, and that he made a profit. Incidentally it is said that the only impurity in the hematite was oxide of copper to the extent of about 25 per cent.

"This, it is said, was the discovery of the first important orebody in the district. This and other mining claims, consolidated into what is now known as the Old Dominion, were bought by Baltimore and Boston people and worked quite extensively considering the fact that the coke had to be brought in from Willcox, 140 miles distant, in wagons, and the black copper pigs had to be hauled back.

"My first visit to the camp was in 1890, when Mr. Douglas began the purchase of claims which later formed the United Globe Mines. At this time Professor Walker was superintendent of the Old Dominion mine A little later some rich and very siliceous and aluminous ores were found in the region near what is now the Miami camp. The Black Warrior shipped rich chrysocolla ore to the Old Dominion smelter in 1895, and later the Keystone and Live Oak, which were located in about 1897, began to ship even more siliceous material to the same works.

"The railway from Bowie, several years in building, reached Globe in the latter part of 1898.

"In 1903 the Old Dominion Copper Mining & Smelting Co. of New Jersey was practically bankrupt. The Old Dominion Copper Mining & Smelting Co. of Maine was formed, and acquired most of the stock of the New Jersey company and all of the stock of the United Globe Mines. In exchange for the latter stock, and with treasury stock bought for cash, gentlemen associated with Phelps, Dodge & Co. took control. A more liberal policy was adopted, profits for the time being were put back into the mine and works, and in four years' time the stock rose from less than $5 to over $60 per share. Ever since that time this mine has been prosperous and profitable, and in your visit you noticed the splendid shape it is in and the great ability of the management.

"Returning to the new district near Miami, in 1903 and 1904, Mr. Coplen, in behalf of himself and others, bonded and purchased a few mining claims which became the beginning of the Inspiration group, and he opened up a small block of lean disseminated ore. In 1907, Mr. J. Parke Channing bonded the Miami group of claims, and developed for himself and his principals the great Miami mine.

"Shortly after, others purchased the Coplen property and many other claims and formed it into the Inspiration Copper Co. and started development on a considerable scale. Hovland and Smith also took over the Live Oak group of claims and like-

wise began development, while parties affiliated with the Miami shareholders took over the Keystone group. The Miami company, after developing their mine, undertook the construction of their splendid plant, which up to that date excelled anything that had ever been built. As it was necessary for them to sell their concentrates they made a long-time contract with Cananea on very favorable terms. A little later, in 1911 as I recall it, the International Smelting & Refining Co. closed a long-time contract for Inspiration concentrates, and later on interests affiliated with this company purchased the Live Oak group and consolidated with the Inspiration Copper Co. into what is known as Inspiration Consolidated Copper Co. This consolidation occurred in March, 1912, and Mr. C. E. Mills accepted the management of the property.

"At first it was planned to use gravity concentration, and construction was begun on a building designed for that process. Late in 1912, however, Dr. Gregory of the Mineral Separations company advised us that his firm had had splendid results on our ore in their laboratory by flotation. It was therefore arranged that he was to send us a 50-ton machine to Inspiration. Dr. Gahl's most able paper tells the rest of the flotation story.

"After taking the two long-term contracts from Miami and Inspiration, it became evident to the companies interested that the high freight rates to Cananea made it

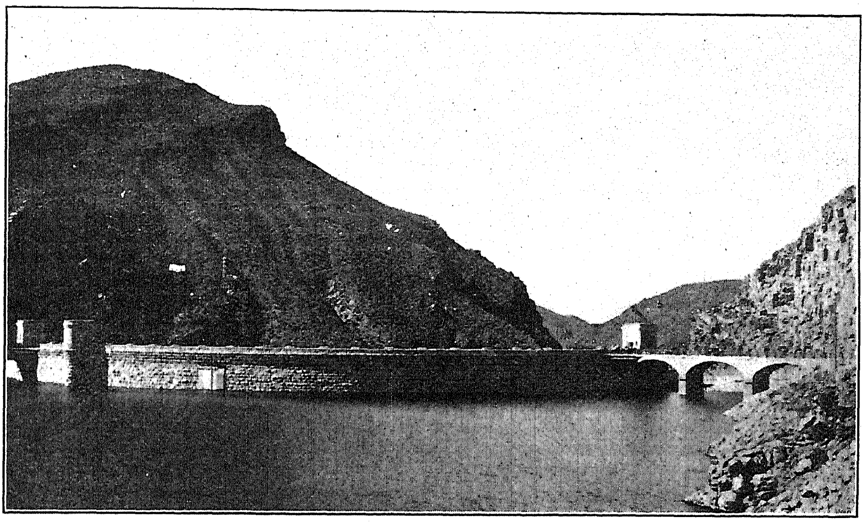


Fig. 6.—Roosevelt Dam from Above.

desirable, and the International smelter now in operation at Miami was decided upon and built.

"In 10 short years the output of the Globe district has sprung from about 30,000,000 lb. of copper a year to about 230,000,000 lb. a year, and the works you have seen today have been put in operation.

"In closing this history, gentlemen, I cannot help telling an anecdote. Both Mills and I once worked for Phelps, Dodge & Co. for many years. For some reason many believe that I still work for them, though I left those good friends 10 years ago. Today I met a gentleman who knew us both in the old days and we were looking over the splendid work at Inspiration, when he said, 'What I can't understand is, how did you let Mills get away from you?' Gentlemen, I had not overlooked a bet. Referring to the puns of a newspaper cartoonist, I had early felt that Mr. Mills was indubitably to be the 'spir' in Inspiration.

"And now, gentlemen, our trip is wellnigh over. We have seen all of the black copper pigs we are going to see, and I feel sure you feel satisfied and rewarded for your long, arduous trip. In leaving Globe, therefore, I feel that it is in order and fitting that I, as your president, should tender our thanks to our hosts, the executiive committee, the local committees, and the individual members of the Arizona Section, which covers, I understand, New Mexico and El Paso members."

At an early hour on Saturday morning, Sept. 23, the Institute party started in automobiles over the famous Apache Trail, which leads for about 50 miles over the plains to the Roosevelt Dam, passing en route cliff dwellings which can be plainly seen from the road. From the Roosevelt Dam, which was an object of considerable interest to all the engineers, to the City of Phoenix, a distance of about 70 miles, the road travels along the gorge of the Salt River and then ascends to a mesa through heavily eroded gorges which form some of the most spectacular scenery in Arizona. The road itself is a real engineering feat, having been built by the Government for the purpose of carrying in supplies for the building of the Roosevelt Dam. As the party emerged from the mountains and came out into the Western Plain with the sun approaching the horizon, the scenery was indescribably beautiful.

Several hours were spent in Phoenix visiting with local members and partaking of a dinner which had been arranged for the visitors. The special train had been brought around by rail and late in the evening the party started for the Grand Canyon, where all arrived several hours late, but still very happy, early Sunday afternoon. The remainder of Sunday and Monday were spent visiting different points on the rim or on the trip down the Canyon. The party was now considerably diminished in number, various members having left for different points of the compass, so that only three special cars departed for the East on Monday night. Owing to a derailment to the west of Williams, the cars were held at that town until early Tuesday afternoon, but the waiting time was spent in an exciting game of baseball, and no one seemed to regret delaying the time for bringing to a close the 113th Meeting of the Institute.

The following members and guests registered at different points during the meeting, but the list is known to be incomplete as a good many attended different functions without registering:

L. M. Allen, Globe, Ariz.
R. S. Allen, Globe, Ariz.
Mrs. R. S. Allen, Globe, Ariz.
J. Owen Ambler, Douglas, Ariz.
J. J. Ambrose, Hayden, Ariz.
F. T. Anderson, El Paso, Tex
C. E. Arnold, Miami, Ariz.
Frank Ayer, Miami, Ariz.
Percy E. Barbour, New York, N. Y.
I. H. Barkdoll, Globe, Ariz.
Mrs. I. H. Barkdoll, Globe, Ariz.
George D. Barron, Rye, N. Y.
Mrs George D. Barron, Rye, N. Y.
P. G. Beckett, Globe, Ariz.
A. D. Beers, New York, N. Y.
Fred. B. Ely, Superior, Ariz.
Allen T. Bird, Nogales, Ariz.
G. N. Bjorge, Globe, Ariz.
L. A. Blackner, Ray, Ariz.
F. C. Blickensderfer, Morenci, Ariz.
A. L. Bloomfield, Denver, Colo.
W. B. Boggs, New York, N. Y.
Mrs. M. Bookman, St. Louis, Mo.
H. P. Bowen, Miami, Ariz.
Mrs. H. P. Bowen, Miami, Ariz.
R. R. Boyd, Globe, Ariz.
M. L. Bradt, Saginaw, Mich.
S. D. Bridge, Comfort, Tex.
C. J. Briggs, El Paso, Tex.
J. F. Brown, Goldfield, Nev.
W. C. Browning, Superior, Ariz.
D. W. Brunton, Denver, Colo.
H. A. Buehler, Rolla, Mo.
Edward E. Bugbee, Boston, Mass.
W. Burns, Morenci, Ariz.
A. B. Calhoun, Globe, Ariz.
K. P. Campbell, Sasco, Ariz.
Norman Carmichael, Clifton, Ariz.
Charles A. Chase, Denver, Colo.
Will L. Clark, Jerome, Ariz.
W. M. Claypool, Needles, Cal.
Ben H. Cody, Clifton, Ariz.
Carl H. Cole, Douglas, Ariz.
David Cole, El Paso, Tex.
Mrs. David Cole, El Paso, Tex.
G. M. Colvocoresses, Humboldt, Ariz.
H. Cooper, El Paso, Tex.
F. G. Cottrell, Washington, D. C.
W. B. Cramer, Globe, Ariz.
Arthur Crowfoot, Morenci, Ariz.
Joseph F. Cullen, Midvale, Utah.
Arthur C. Daman, Denver, Colo.
S. H. Davis, Joplin, Mo.
E. G. Deane, Miami, Ariz.
George C. Dewey, Selby, Cal.
R. H. Dickson, Warren, Ariz.
Albert Doerr, South Pasadena, Cal.
Kuno Doerr, El Paso, Tex.

S. G. Dolman, Ray, Ariz.
Thos. F. Donnelly, Tucson, Ariz.
Walter Douglas, Bisbee, Ariz.
W. M. Drury, El Paso, Tex.
R. G. Dufourcq, El Paso, Tex.
H. S. Duncan, Globe, Ariz.
H. F. Durke, El Paso, Tex.
A. V. Dye, Douglas, Ariz.
Howard Eckfeldt, S. Bethlehem, Pa
J. A. Ede, La Salle, Ill.
Karl Eilers, New York, N. Y.
M. J. Elsing, Cananea, Son., Mex.
Mrs. M. J. Elsing, Cananea, Son., Mex.
M. Elsasser, Los Angeles, Cal.
C. T. Emrich, Globe, Ariz.
Mrs. C. T. Emrich, Globe, Ariz.
E. N. Engelhardt, Oakland, Cal.
I. E. Ettinger, Superior, Ariz.
Robert Faulkner, Lebanon, Pa.
Percy LeR. Fearn, New York, N. Y.
Leon Feuchère, Warren, Ariz.
Siegfried Fischer, Jr., Golden, Colo.
H. A. Fitch, Kansas City, Mo.
F. N. Flynn, Clifton, Ariz.
J. G. Flynn, Miami, Ariz.
Paul R. Forbes, New York, N. Y.
Robert Franke, Miami, Ariz.
W. E Gaby, Butte, Mont.
Rudolf Gahl, Miami, Ariz.
Mrs. Rudolf Gahl, Miami, Ariz.
W. I. Garms, Hayden, Ariz.
W. F. Geiger, Miami, Ariz.
R. J. Glendinning, Salt Lake City, Utah.
W. B. Gohring, Warren, Ariz.
C. W. Goodale, Butte, Mont.
William D. Gordon, El Paso, Tex.
B. Britton Gottsberger, Miami, Ariz.
Mrs. B. Britton Gottsberger, Miami, Ariz.
C. A. Grabill, El Paso, Tex.
J. C. Greenway, Warren, Ariz.
Alden D. Groff, New York, N. Y.
Walter Gross, Perth Amboy, N. J.
Justice Grugan, Edwards, N. Y.
R. Dawson Hall, New York. N. Y.
W. S. Hall, Miami, Ariz.
Herbert E. Hambleton, El Paso, Tex.
James W. Hambleton, El. Paso, Tex.
A. M. Hamilton, Sasco, Ariz.
A. M. L. Hamilton, Sasco, Ariz.
H. T. Hamilton, Nacozari, Mex.
H. O. Hammond, Douglas, Ariz.
R. S Handy, Kellogg, Idaho
A. B. Hardie, Philadelphia, Pa.
E. Harms, Torreon, Mex.
Walter Harris, Globe, Ariz.
Ruger W. Hay, Warren, Ariz.
Justin H. Haynes, Denver, Colo.
Stewart Hazelwood, San Francisco, Cal.
Jas. L. Head, Warren, Ariz.
Miss C. Hegeler, Danville, Ill.
Edward C. Hegeler, Danville, Ill.
Mrs. Edward C. Hegeler, Danville, Ill.
Julius W. Hegeler, Danville, Ill.
Murray Hendrix, Miami, Ariz.
J. H. Hensley, Jr., Miami, Ariz.
E. C. Hickman, East Helena, Mont.
F. G. Hills, Leadville, Colo.
F. W. Hoar, Globe, Ariz.
E. N. Hobart, Nogales, Ariz.
J. P. Hodgson, Bisbee, Ariz.
C. E. Hogue, Globe, Ariz.
Mrs. C. E. Hogue, Globe, Ariz.
L. O. Howard, Globe, Ariz.
M. R. Hull, Clifton, Ariz.
H. D. Hunt,
Mrs. H. D. Hunt,
Alexander Imhoff, Los Angeles, Cal
T. H. Jenks, Wickenburg, Ariz.
S. J. Jennings, New York, N. Y.
Mrs. S. J. Jennings, New York, N. Y.
Miss Amy S. Jennings, New York, N. Y
Miss Mary A. Jennings, New York, N. Y.
Frank E. Johnson, Salt Lake City, Utah
William Strickler Jones, Atlantic City N. J.
Ira B. Joralemon, Warren, Ariz.
D. N. Kay, Ray, Ariz.
G. W. Kays, Kingman, Ariz.
R. W. Kerns, Warren, Ariz.
S. J. Kidder, Mogollon, New Mex.
R. B. T. Kiliani, New York, N. Y.
K. L. Kithil, Tucson, Ariz.
Edward H. Koenig, Perth Amboy, N. J.
A. S. Konselman, Cananea, Mex.
J. Kruttschnitt, Jr., Tucson, Ariz.
O. M. Kuchs, Tooele, Utah.
C. R. Kuzell, Anaconda, Mont.
John Langton, New York, N. Y.
E. H. Laws, Salida, Colo.
C. Legrand, Douglas, Ariz.
S. H. Levison, Hayden, Ariz.
W. W. Logue, Hayden, Ariz.
B. M. McAtee, Miami, Ariz.
Alan F. McCormick, El Paso, Tex.
W. E. McCourt, St. Louis, Mo.
Geo. T. McGee, Helena, Mont.
A. G. McGregor, Bisbee, Ariz.
P. M. McHugh, Denver, Colo.
R. McIntosh, Lake Linden, Mich.
Mrs. R. McIntosh, Lake Linden, Mich.
Will E. McKee, Bisbee, Ariz.
William T. MacDonald, Hayden, Ariz.
F. W. Maclennan, Miami, Ariz.
J. F. Manning, Holkol, Korea.
E. R. Marble, Hayden, Ariz.
Emory M. Marshall, Globe, Ariz.
Mrs. Emory Marshall, Globe, Ariz.
E. P. Mathewson, Anaconda, Mont.
Mrs. E. P. Mathewson, Anaconda, Mont.
E. V. Matlack, Webster Grove, Mo.
Ellwood V. Matlack, Jr., Webster Grove, Mo.
M. S. Mazany, Miami, Ariz.
A. H. Means, Tucson, Ariz.
H. I. Merricks, Miami, Ariz.
C. W. Merrill, San Francisco, Cal.
Mrs. C. W. Merrill, San Francisco, Cal.
F. J. H. Merrill, Los Angeles, Cal.

C. E. Mills, Globe, Ariz.
Edwin W. Mills, Holkol, Korea.
Charles A. Mitke, Bisbee, Ariz.
Philip N. Moore, St. Louis, Mo.
H. W. Morse, Los Angeles, Cal.
McHenry Mosier, Bisbee, Ariz.
P. A. Mosman, New York, N. Y.
Seeley W. Mudd, Los Angeles, Cal.
H. T. Murray, Hayden, Ariz.
R. T. Murrill, Flat River, Mo
Henry W. Nichols, Chicago, Ill.
H. L. Norton, Globe, Ariz.
Arthur Notman, Bisbee, Ariz.
T. H. O'Brien, Dawson, N. Mex.
J. J. Ormsbee, El Paso, Tex.
H. D. Pallister, El Paso, Tex.
J. H. Payne, New York, N. Y.
Mrs. J. H. Payne, New York, N. Y.
E. F. Pelton, Tyrone, N. Mex.
Basil Prescott, El Paso, Tex.
R. J. Pritchard, Miami, Ariz.
William J. Quigly, El Paso, Tex.
O. C. Ralston, Salt Lake City, Utah.
F. L. Ransome, Washington, D. C.
Stuart L. Rawlings, San Francisco, Cal.
William H. Rea, Pittsburgh, Pa.
Mrs. Wm. H. Rea, Pittsburgh, Pa.
Miss Rea, Pittsburgh, Pa.
E. E. Reyer, El Paso, Tex.
C. E. Rhodes, Pasadena, Cal.
Miss Marion Rice, Schenectady, N. Y.
Walter A. Richelsen, Cananea, Sonora, Mex.
L. D. Ricketts, Warren, Ariz.
Mrs. L. D. Ricketts, Warren, Ariz.
Ezra B. Rider, Bisbee, Ariz.
A. E. Ring, Flat River, Mo.
J F. Robertson, Coniston, Ont.
Burr A. Robinson, New York, N. Y.
E. M. Robinson, So. Bethlehem, Pa.
Clyde P. Ross, Globe, Ariz.
E. W. Rouse, Baltimore, Md.
G. H. Ruggles, Miami, Ariz.
J. P. Ruminapp, Miami, Ariz.
B. E. Russell, Ray, Ariz.
F. Rutherford, Douglas, Ariz.
E. M. Sawyer, Tyrone, N. M.
Francis S. Schimerka, Clifton, Ariz.
Walter A. Schmidt, Los Angeles, Cal.
Carl Scholz, Chicago, Ill.
C. D. Schultz,
W. Schumacher, El Paso, Tex.
Mortimer A. Sears, Santa Fe, New Mex
Gerald Sherman, Bisbee, Ariz
L. B. Shipley, New York, N. Y.
Alex Sibbald, Globe, Ariz.
H. R. Simpson, Los Angeles, Cal
K. M. Simpson, Los Angeles, Cal.
Howard D. Smith, San Francisco, Cal.
E. G. Snedaker, Goldfield, Nev.
F. W. Solomon, Miami, Ariz
P. G. Spilsbury, New York, N. Y.
E. M. Steele, Hayden, Ariz.
P A. Steger, Miami, Ariz.
Paul Stein, El Paso, Tex.
Paul Sterling, Wilkes-Barre, Pa
E. D. Stewart, El Paso, Tex
Bradley Stoughton, New York, N. Y.
Mrs. Bradley Stoughton, New York, N. Y.
Roger Strobel, East Helena, Mont.
Wilmer C. Swartley, Philadelphia, Pa.
W. S. Sultan, Globe, Ariz.
John C. Taylor, Denver, Colo.
Knox Taylor, High Bridge, N. J.
Roscoe Teats, Tacoma, Wash.
B. B. Thayer, New York, N. Y.
O. J. Tuschka, Globe, Ariz.
Mrs. O. J. Tuschka, Globe, Ariz.
G. D. Van Arsdale, New York, N. Y.
R. E. Vining, Perth Amboy, N. J.
John D. Wanvig, Golconda, Ariz.
Harry S. Ware, Anaconda, Mont.
J. H. Watkins, Washington, D. C.
Arthur P. Watt, St. Francois, Mo.
A J. Weinig, Telluride, Colo
Harry V. Welch, Los Angeles, Cal.
George H. West, Globe, Ariz.
J. R. Wester, Morenci, Ariz.
Charles H. White, Cambridge, Mass.
J. L. White, Humboldt, Ariz.
H. E. Williams, Calumet, Mich.
Mrs. H. E. Williams, Calumet, Mich.
J. S. Williams, Nacozari, Mex.
Bailey Willis, Stanford Univ., Cal.
Philip D. Wilson, Warren, Ariz.
Philip Wiseman, Los Angeles, Cal.
Mrs. Philip Wiseman, Los Angeles, Cal.
C. L. Wolfle, El Paso, Tex.
J. R. Woodul, Mexico.
Wm H. Yeandle, Jr.
R. B. Yerxa, Miami, Ariz.
H. M. Ziesemer, Bisbee, Ariz.

Technical Sessions

The full list of papers which were presented by their authors or by title at the meeting is as follows:

Mining

Mine Accounting for Small Mines. By James E. Chapman.
Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co. By H. Kenyon Burch and M. A. Whiting.

Geology and Mineralogy

Cyanidation

Miscellaneous

Flotation and Ore Dressing

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* Held for publication in Vol. LVI.

A NEW SOURCE OF FLOTATIVE AGENTS. By G. H. Clevenger.
HISTORY OF THE FLOTATION PROCESS AT INSPIRATION. By Rudolf Gahl.
SOME MISCELLANEOUS WOOD OILS FOR FLOTATION. By R. C. Palmer, Glenn L. Allen and O. C. Ralston.
THE ADVENT OF FLOTATION IN THE CLIFTON-MORENCI DISTRICT, ARIZONA. By David Cole.
A COMBINED HYDRAULIC AND MECHANICAL CLASSIFIER By M. G. F. Sohnlein.
COMPARATIVE TEST OF THE MARATHON, CHILEAN AND HARDINGE MILLS. By F. C Blickensderfer.
MINE AND MILL PLANT OF THE INSPIRATION CONSOLIDATED COPPER CO. By H. Kenyon Burch.

Smelting, Etc.

THE DECOMPOSITION AND REDUCTION OF LEAD SULPHATE AT ELEVATED TEMPERATURES. By W. Mostowitsch. Edited by H. O. Hofman.
DETERMINATION OF DUST LOSSES AT THE COPPER QUEEN REDUCTION WORKS. By J. Moore Samuel.
AN INVESTIGATION INTO THE FLOWING TEMPERATURES OF COPPER MATTES AND OF COPPER-NICKEL MATTES. By G. A. Guess and F. E. Lathe.
FEATURES OF THE NEW COPPER SMELTING PLANTS IN ARIZONA. By A. G. McGregor.
SMELTING AT THE ARIZONA COPPER CO.'S WORKS. By F. N. Flynn.
THE BASIC LINED CONVERTER IN THE SOUTHWEST. By L. O. Howard.

Leaching

2,000-TON LEACHING PLANT AT ANACONDA. By Frederick Laist and Harold W. Aldrich.
POSSIBILITIES IN THE WET TREATMENT OF COPPER CONCENTRATES. By Lawrence Addicks.
LEACHING TESTS AT NEW CORNELIA. By H. W. Morse and H. A. Tobelmann.

Ore Deposits

GOLD AND SILVER DEPOSITS OF NORTH AND SOUTH AMERICA. By Waldemar Lindgren.
*FUEL IN TURKEY. By Leon Dominian.
*MANGANESE ORES OF RUSSIA, INDIA, BRAZIL AND CHILE. By E. C. Harder.
THE EMERALD DEPOSITS OF MUZO, COLOMBIA. By Joseph E. Pogue.
THE RADIO-ACTIVITY OF ALLANITE. By L. S. Pratt.
ZIRCON-BEARING PEGMATITES IN VIRGINIA. By Thomas L. Watson.
IRON PYRITES DEPOSITS IN SOUTHEASTERN ONTARIO, CANADA. By P. E. Hopkins.

Petroleum and Gas

*PRINCIPLES OF NATURAL GAS LEASEHOLD VALUATION. By S. S. Wyer.
*THE CALIFORNIA GASOLINE INDUSTRY. By W. R. Hamilton.
*THE DIASTROPHIC THEORY. By Marcel R. Daly.
*THE POSSIBILITY OF DEEP SAND OIL AND GAS IN THE APPALACHIAN GEO-SYNCLINE OF WEST VIRGINIA. By David B. Reger.

Instead of attempting to present all of these papers for discussion, the Board of Directors, upon the recommendation of the Committee on Papers and Publications, selected those papers which could be adequately presented by their authors or authors' representatives and which lent themselves particularly to discussion. Six technical sessions were then established and each was devoted to a particular subject with the understanding that the appropriate papers would first be read and discussed and following that general discussion of the subject would be welcomed. Opportunity was also given at every session for any person present to call for any paper on the list to be brought up for discussion.

* Held for publication in Vol. LVI.

The first technical session was devoted to the subject of "Smelting" and was held at the Y. M. C. A. building, Douglas, Ariz., Tuesday afternoon, Sept. 19, at 2 o'clock.

President Ricketts announced that the 113th Meeting of the Institute was opened and called upon Walter Douglas to preside. The presiding officer then called upon John C. Greenway, who delivered a cordial Address of Welcome to Arizona. In response, President Ricketts announced that as he was both guest and host at the Arizona Meeting, he would call upon Past President Benjamin B. Thayer, who thereupon responded cordially to the Address of Welcome.

The following papers were then presented by their authors:

FEATURES OF THE NEW COPPER SMELTING PLANTS IN ARIZONA. By A. G. McGregor. Discussed by L. D. Ricketts, E. P. Mathewson.

SMELTER OPERATING NOTES FROM THE ARIZONA COPPER CO., LTD. By F. N. Flynn.

THE BASIC LINED CONVERTER IN THE SOUTHWEST. By L. O. Howard. Discussed by Walter Douglas, E. P. Mathewson, Kuno Doerr.

DETERMINATION OF DUST LOSSES AT THE COPPER QUEEN REDUCTION WORKS. By J. Moore Samuel. Discussed by Walter Douglas, E. P. Mathewson, S. J. Jennings, C. E. Arnold, A. G. McGregor, L. D. Ricketts.

The second technical session was on the subject of "Leaching" and was held at the Y. M. C. A. building at Douglas, Ariz., on Tuesday evening, Sept. 19, at 8 o'clock. H. W. Morse presided.

President Ricketts opened the meeting by reading the following telegram from Dr. Douglas, which was greeted with applause:

"Please convey my hearty greetings to the Institute and my regrets that my health prevents my presence with you in person. It is nearly 17 years since we met around the dinner table in Bisbee and lunched together in the mine. Though we cannot anticipate eternal life for our mines you will be glad to find that it is more productive than it was and is enjoying a vigorous old age and renewing its youth by new discoveries."

and the President was authorized to respond to this in the name of the members present, as follows:

"The members of the Institute at this meeting, some two hundred and fifty in number, by special resolution unite with me in thanking you for your most welcome message of good-will. We owe so much to you and it is good to feel we have your thoughts and good wishes. We thank you and send you our greetings and best wishes for you and yours."

The following papers were then presented:

LEACHING TESTS AT NEW CORNELIA By H. W. Morse and H. A. Tobelmann. (Presented by H W. Morse.) (Discussed by G. D. Van Arsdale, S. J. Jennings, F. S. Schimerka, C. G. Grabill, F. N. Flynn, written discussion by Lawrence Addicks.)

2,000-TON LEACHING PLANT AT ANACONDA. By Frederick Laist and Harold W. Aldrich. (Presented by E. P. MATHEWSON.) (Discussed by F. N. Flynn, E. P. Mathewson.)

POSSIBILITIES IN THE WET TREATMENT OF COPPER CONCENTRATES. By Lawrence Addicks. (Presented by the Secretary.) (Discussed by F. N. Flynn.)

GENERAL SUBJECT OF LEACHING was discussed by F. S Schimerka, B. B. Gottsberger, H. E. Williams, G. D. Van Arsdale, and E. P. Mathewson.

The third technical session was on the subject of "Mining and Geology" and was held in the auditorium of the high school at Bisbee, Ariz., on the afternoon of Wednesday, Sept. 20, at 2 o'clock. Gerald F. G. Sherman presided.

The following papers were presented by their authors:

PETROGRAPHY OF THE MOUNT MORGAN MINE, QUEENSLAND. By Walter E. Gaby. (Discussed by L. C. Graton.)
GEOLOGY OF THE WARREN MINING DISTRICT. By Y. S. Bonillas, J. B. Tenney, Leon Feuchère. (Presented by Y. S Bonillas.) (Discussed by I. B. Joralemon, F. L. Ransome, Gerald Sherman, L. C. Graton.)
STOPING IN THE CALUMET AND ARIZONA MINES, BISBEE, ARIZ. By Philip D. Wilson.
COÖPERATIVE EFFORT IN MINING. By Joseph P. Hodgson. (Discussed by J. A. Ede, Gerald F. G. Sherman, C. W. Goodale, Charles A. Mitke, C. E. Arnold, D. W. Brunton.)

The fourth technical session was on the subject of "Concentration and Flotation" and was held at the Martin Theater, Globe, Ariz., on Thursday afternoon, Sept. 21, at 2 o'clock. C. E. Mills presided.

The following papers were presented:

THE ADVENT OF FLOTATION IN THE CLIFTON–MORENCI DISTRICT, ARIZONA. By David Cole.
HISTORY OF THE FLOTATION PROCESS AT INSPIRATION. By Rudolf Gahl. (Discussed by F. S. Schimerka, H. W. Morse, R. S. Handy, C. A. Chase, E. P Mathewson, David Cole, L. D. Ricketts, G. H. Ruggles, F. G. Cottrell; written discussion by Frederick Laist and R. C. Canby.)
SOME MISCELLANEOUS WOOD OILS FOR FLOTATION. By R. C Palmer, Glenn L. Allen and O. C. Ralston. (Presented by G. L. Allen.)
FLOTATION CONCENTRATION AT ANACONDA, MONT. By Frederick Laist and Albert E. Wiggin. (Presented by E. P. Mathewson.) (Discussed by O. C. Ralston, David Cole, E. P. Mathewson, Rudolf Gahl, Norman Carmichael, W. B. Cramer, C. W. Merrill, R. S. Handy, A. P. Watt, B. B. Gottsberger, F S. Schimerka.)

The fifth technical session was on the subject of "Mining" and was held at the Martin Theater, Globe, Ariz., on the evening of Sept. 21, at 8 o'clock. Percy G. Beckett presided.

The following papers were presented:

THE BLOCK METHOD OF TOP SLICING OF THE MIAMI COPPER CO. By E. G. Deane. (Discussed by J. A. Ede, J. P. Hodgson.)
STOPING METHODS OF MIAMI COPPER CO. By David B. Scott. (Presented by the Secretary.)
MINE FIRE METHODS EMPLOYED BY THE UNITED VERDE COPPER CO. By Robert E. Tally. (Presented by the Secretary.) (Written discussion by C. L. Berrien.) (Discussed by C. W. Goodale, J. A Ede, Gerald Sherman, J. P. Hodgson.)
ORE DRAWING TESTS AND THE RESULTING MINING METHOD OF INSPIRATION CONSOLIDATED COPPER CO. By George R. Lehman.
COST AND EXTRACTION IN THE SELECTION OF A MINING METHOD. By C. E. Arnold. (Discussed by L. D Ricketts, F. W. Maclennan.)

After the papers scheduled for the meeting had been presented, Walter A. Schmidt gave a brief abstract of a paper prepared by himself, entitled "Results Obtained with the Cottrell Process at Smelter of International Smelting Co., Inspiration, Ariz. and the Factory of Riverside Portland Cement Co., Riverside, Cal.,"* illustrating his remarks with stereopticon slides, and ending with a demonstration of the process.

The sixth technical session was on the subject of "Fine Grinding" and was held at the Club House of the Miami Copper Co., Miami, Ariz., on the afternoon of Friday, Sept. 22, at 4 o'clock. B. B. Gottsberger presided.

The following papers were presented by their authors:

POWER PLANT OF BURRO MOUNTAIN COPPER CO. By Charles Legrand. (Discussed by B. B. Gottsberger, S. J. Kidder, and John Langton.)
COMPARATIVE TEST OF THE MARATHON, CHILEAN AND HARDINGE MILLS. By F. C. Blickensderfer. (Discussed by B. B. Gottsberger, R. B. Yerxa, A. P. Watt, R. B. T. Kiliani, F. S. Schimerka, S. J. Jennings, R. S. Handy, C. W. Merrill, Robert Franke, F. J. H. Merrill.)

* Not published.

PAPERS

Mine Accounting for Small Mines*

BY JAMES E. CHAPMAN, WALLACE, IDAHO

(Arizona Meeting, September, 1916)

THE observations here presented are not those of an expert accountant, but of one who, while he has seen considerable service in the accounting departments of large companies, has spent more time in engineering and operating.

This paper is intended to cover, in a measure, mine accounting for small mines, as distinguished from the elaborate systems, requiring many persons in the accounting department. I shall attempt to outline a system embracing the essentials of accounting, and simple enough in form to permit one or two persons to carry it on from month to month, in sufficient detail to be able to tell quickly the grade of ore, the prices received for metals, costs per ton for mining and milling, costs per foot for development, upward or downward tendencies in costs, ore settled or in transit, cash on hand, stocks of supplies on hand, efficiency of labor, etc.

As in all accounting, there are two main divisions: that of revenue received for what is sold, and that of disbursements made for what is bought, so in mine accounting we have to consider chiefly the income derived from sales of ore or concentrates, and the expenses incurred in producing the said ore or concentrates in a marketable condition.

I. *Income*

The revenue of a mining company from the sale of ore or concentrates comes in the form of remittances, accompanied by settlement sheets made out by the buyer of the mine product, such as a smelting company. It is the duty of the mine accountant (or the person to whom this is assigned in connection with other work) to check these settlement sheets, against both the mine weights and assays, and the schedules of prices arranged between the mine and the smelter. As a rule, quotations for silver, lead, zinc, copper and other metals enter into the settlements, and require checking, as do also bills of lading. Each step of the arithmetical calculation is also checked.

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* Originally presented at the Wallace, Idaho, Meeting of the Columbia Section, Nov. 19, 1915.

The essential figures from these sheets are entered in an Ore Record, in columns headed "Wet Tons," "Moisture Per Cent," "Moisture Tons," "Dry Tons," "Assays per Ton," "Prices per Ton," "Contents of Metals" (in ounces and pounds), "Gross Value," "Smelter Freight and Treatment," "Net Value," and possibly others, the entries being segregated by lots, and classes of ore. Stocks on hand unsettled and in transit at the end of the month are added to the settlement figures, mine assays and estimates being used to arrive at an approximate value, and from these totals are deducted similar estimates of ore unsettled and in transit at the beginning of the month. The result is the month's production, and its net value is posted to the Ledger, to the credit of ore account and the debit of the smelting company, or to the debit of head office, in case the smelter makes its payments to such an office, on which the mine draws for funds, as required to pay operating expenses.

Secondary records in the form of a shipment book, files of bills of lading, assay certificates, etc., may be kept as the accountant finds most convenient, and according to the local conditions and circumstances of the property. Their purpose is primarily to facilitate the checking of the settlements and to aid in an accurate estimate of the amount, grade and value of ore awaiting settlement.

"Miscellaneous income" is a term which will apply to all other revenue, whether from the rent of houses owned by the mining company, or the profit on the company store or boarding house, or interest, or exchange, or dividends, or the sale of junk, tailings, etc. These are usually cash items, and are taken care of in the Cash and Voucher Record, which will now be considered.

The Cash and Voucher Record, as a single book, preferably looseleaf, has met with general approval in recent years as a combination of cashbook, journal, and voucher register, and is the only record at the bookkeeper's desk from which posting is done to the ledger. It is compact, convenient, and desirable, serving the purpose of gathering numerous items into their proper sections at original entry, and requiring that they balance before they are used in further calculations. This book may have columns for cash, bank accounts, operating accounts, and various non-operating ledger accounts. It seems scarcely necessary to do more than sketch the use of this record, since its cash columns correspond exactly to those of a cashbook, and journal entries are made in it double, as in a journal, the only difference being that the amounts are listed in columns so that they are easier to refer to and total. Only totals are posted to the ledger, so that posting is a short and simple matter. This is done but once a month, after closing the month's operating accounts, and the purpose is to prove the balance between assets and liabilities.

II. *Operating Expenses*

Next to profits, the figures of greatest interest to a mine operator are doubtless the operating costs. The two, profits and costs, may almost be said to be complementary, and many mine operators do not doubt that if they hammer down their costs, their profits will go up. But it is capable of demonstration that an additional expenditure per ton may, under certain conditions, increase the grade of smelting ore or mill feed to a point where profits will be increased by more than the extra outlay.

For the measurement of efficiencies and the planning of improvements, a study of costs is necessary. The record of these, as often kept, not only has no statistical value, but may even be misleading, so that when work is based upon the data they furnish an expected saving will show up as a loss instead. This is particularly true of development work. Sinking a shaft, for instance, is an operation into which many factors enter to determine the true cost per foot of depth. Some of these factors, often not considered at all when a contract is let, are: The cost of air from compressor; the proportion of hoisting, pumping and timbering expense; labor and fuel for sharpening steel; repair parts and time spent in repairing machines.

An accounting system, to have value for the mine owner, should be arranged so that the important figures for use in estimating the cost of a certain operation, such as the above, can be taken off without undue waste of time, and also so that the upward or downward trend of the costs per ton in any or all departments can be seen at a glance. Practically, each mine needs to have its accounting method, almost as much as its plan of development, adjusted to its circumstances.

In this connection, I propose to discuss briefly two or three systems of classification of operating costs, evolved by large companies with whose methods I am more or less familiar, and to point out their adaptability in a measure to the requirements of smaller companies, whose office staffs consist of fewer persons, and whose need for detail, beyond the necessary working figures, is smaller.

In the first place, the distinction is drawn between direct and indirect operating costs, direct costs being, as the term indicates, those incurred for actual handling of ore, and indirect those necessitated by the operation of mining in general.

One of the largest operating companies has the following accounts for its smaller properties:

1. *Direct.*—(*a*) Development; (*b*) Ore Breaking and Stoping; (*c*) Timbering; (*d*) Tramming; (*e*) Hoisting; (*f*) Sorting, Weighing and Loading; (*g*) Draining.

2. *Indirect.*—(*h*) Branch Office; (*i*) Salaries; (*j*) General and New

York Expense; (*k*) Insurance; (*l*) Taxes; (*m*) Laboratory; (*n*) Marketing; (*o*) Transportation.

At larger and more complicated properties, the expenses are distributed on this outline, with certain modifications and amplifications. Ore Breaking and Stoping may have subdivisions as follows: Drill Steel; Machine Drills, Repairs and Maintenance; Mining Tools; Explosives; Lighting; and Pipe Lines and Tracks (Pipe lines for air and ventilation, and Track for tramming in stopes). Timbering may have subdivisions such as: Tools; Stulls, Lagging, etc.; and Miscellaneous. Tramming, which, as an account, includes only tramming on levels, may be subdivided as follows: Track and Fittings; Car and Motor Repairs and Maintenance; Animals and Feed; and Miscellaneous. Hoisting, which includes tramming from shaft-collar through an adit in the case of an interior shaft, may be divided into: (*a*) *Shaft.* Equipment Repairs and Maintenance; Shaft Repairs and Maintenance; and Miscellaneous. (*b*) *Adit.* Track and Fittings; Car and Motor Repairs and Maintenance; Animals and Feed; and Miscellaneous.

For a concern operating a mill, the number of accounts is increased by the addition of Sampling, Mill Experimental Work, and Milling. Only the last-named requires a further subdivision in order to give more definite information as to the work and the efficiency of the various departments and machines. A good arrangement of such subdivided accounts has been found to be: 1. Crusher Supplies; 2. Roll Supplies; 3. Huntington-Mill Supplies; 4. Hardinge- and Tube-Mill Supplies; 5. Elevator Supplies; 6. Trommel Supplies; 7. Jig, Table and Vanner Supplies; 8. Pump Supplies; 9. General Mill Labor; 10. Loading Concentrates; 11. Labor for Repairs and Maintenance.

"General Expense," if care is not exerted, may become a dumping place for items which ought properly to be distributed to accounts to which they pertain. The heading "Miscellaneous" is practically taboo, except as the designation of a department in which details are shown. Two or three large companies known to me, and probably many others, group all indirect charges under the head of "general" expenses, included in which is a subheading of "general" or "miscellaneous" expenses. For checking up the latter, a memorandum sheet is carried in the Distribution Book, on which spaces are provided for as many as 25 or 30 different items, among them office supplies, subscriptions, postage stamps, revenue stamps, janitor and cleaning, office fuel and heating, office lights and water, engineering instruments, blue-printing supplies, telephones, telegrams, traveling expenses, etc.

One well-known company keeps tally of these elusive items by providing for many of them among the indirect charges. Its list of indirect accounts contains 23 items: Salaries; Mine Foremen and Bosses; Storekeepers and Timekeepers; Watchmen; Stable; Office Supplies;

Rent; Telephone and Telegraph; Traveling; Lighting; Engineering and Surveying; Document Stamps; Employees' Living Quarters; Hospital Contribution and Expense; Fire Insurance; Employees' Liability Insurance; Legal Expenses; Taxes; New York Office; Accident Expenses; Securing Labor; Strike Expense; and Miscellaneous.

It is thus seen that there is considerable opportunity for meeting individual requirements in the naming and arranging of accounts. There is no hard and fast rule in these matters; and the preceding examples are presented merely as having been tried by experience, and having, under certain conditions, given satisfaction.

Vouchers and Entries.—The actual procedure in accounting for expense items comprises the following steps: (1) Making the voucher; (2) entry in cash-voucher record; (3) entry in distribution book; and (4) summary on cost sheet or monthly report.

The first step consists of writing the check for the net amount due on an invoice, using preferably a voucher-check form, on the reverse side of which is written sufficient detail to identify the transaction, together with the distribution of the amount of the check to various accounts.

In the second step, the total of the amount charged to the operating accounts is debited in a column with the title "Operating," and the amount of the check is credited in the proper bank column.

For the third step, a book with numbered columns is required, each number designating a certain account and applying only to that account. The individual figures making up the total operating charge on each voucher are entered, voucher by voucher, in the proper columns, each voucher occupying a line, and each line checking across to a total operating column, of which the total should agree with the total operating column in the cash-voucher record.

For the fourth step, the cost sheet summary, it has been found to be a good method to letter the totals of the operating accounts on tracings ruled in columns, each column representing a month. The account numbers and names can be lettered on the horizontal lines, and additional lines may be devoted to such items as "tons mined," "tons sorted," "tons milled," etc., and the results of dividing these tonnages into the various totals of expense, *i.e.*, the costs per ton, may be also noted. Prints from these tracings make neat monthly reports, very convenient for comparison since the figures for any item of expense, tonnage or cost per ton are shown side by side on the same line, from month to month throughout the year.

A comprehensive survey of operations is completed by making up a financial statement, by adding the totals of income (separating the amounts according to the classes of ore sold, if desired) and deducting the operating expenses therefrom. This is made up and entered as a voucher also, the balance being charged into profit and loss account.

Distribution of Power, Etc.—It may have been noticed that no reference has been made to power, or to mechanical or electrical time and supplies. This has purposely been deferred until the last, and will now be considered as illustrating the idea of redistribution. Following out this idea, all charges to these departments are divided at the end of the month according to the purpose for which the electric current or supplies were used, or the work done. For example, charges for current are pro-rated, according to kilowatt-hours or horsepower, to mine compressors, pumps, locomotives, hoist motors, framing-shed motors, sorting-plant motors, mill motors for driving various machines, etc. In a similar way, mechanics' and electricians' time may be redistributed by the use of cards specifying hours worked on this or that job, and repair parts used on the same. Thus proper charges may be made against the various motors, engines, and machines, cars, track, trolleys, lights—against any part of the plant, in fact, for the benefit of which this diverse work may be done. No hair-splitting is necessary or at all desirable. Each man can make out his own card in a few minutes at the end of each shift, and there is no reason why a high-priced shop-foreman should devote any time to doing the work of a cheaper clerk.

An adaptation of redistribution, making it much simpler and at the same time giving accurate statistics as to power costs for milling, for example, is to have a power-plant account from which mill-power costs and others, depending on local conditions, are redistributed, leaving a net amount as having been expended for mine power.

Purchases and Inventories.—Two other matters, which are closely related, naturally come under the control of the mine accountant, and call for brief reference. I refer to the ordering of supplies, and the keeping of stock and taking of inventories.

As to the first, the management often wishes to take charge of the actual placing of orders; but in many concerns the routine orders are usually handled by the accounting office, which takes care regularly of the checking of prices and the extensions on invoices. All this work is often delegated, in large companies, to a purchasing department, independent of the accounting department.

A lengthy paper might be written concerning the second subject alone. The main idea as regards stocks of supplies is to account for everything purchased, and charge it out as it is used. Two methods are in vogue, and both work well if a diligent and conscientious man is in charge of supplies; conversely, both work miserably if careless or slipshod methods are allowed.

According to one method, everything is charged to stock account when purchased, and is entered in a stock ledger immediately on its delivery at the plant. When an article is issued for use, a requisition is required, specifying the account for which it is to be used, that account being

then charged and stock account being credited. The stock ledger is thus a perpetual inventory, which requires checking by actual inventory only once or twice a year.

The other method of keeping stock provides that the greater portion of purchases are charged directly to the account for the use of which they are intended, thus obviating the use of a stock ledger. Exception is made, however, in many cases, such as explosives, candles, timber, drill steel, tools, feed, hoists and parts, pumps and parts, crushers, rolls, vanners, jigs, elevators, etc., and their parts. In order to get at the true consumption in such cases, monthly inventories of these large items must be taken, the difference between stocks on hand at the first and last of a month, plus the purchases, representing the value of supplies used.

It should be borne in mind, in deciding for one or the other of these methods, that the fluctuation in costs from month to month, using the first method, is slight, because the charges for supplies in a given month are made up from the requisitions of supplies actually used out of stock during that month. Under the second method, on the other hand, the charges for supplies may vary considerably, depending, as they do, directly on the invoices which are paid during the month. The first method permits paying invoices at any time without throwing the cost averages out of joint, whereas by the second method invoices must be paid just as the material is used, and, furthermore, the books covering a month's business must be held open until the invoices representing that month's business are received and paid.

In adopting a system of accounting, a thorough study should be made of the conditions at the mine where it is to be installed, bearing in mind also the needs and desires of the management and the stockholders in the matter of reports. Once a system has been adopted, it should be adhered to religiously, and the seeming advantages of some different system should be well considered and tried out before any change is made.

Automatic Operation of Mine Hoists as Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co.

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ONE of the advantages presented by electric drive in many classes of work is the ease with which the electric motor can be controlled automatically. In a large number of cases certain features of the control are automatic—for example, the rate of acceleration may be limited automatically or the equipment may be stopped automatically at the limit of travel—but the equipment is started and ordinarily is stopped by hand. In other cases the motion of the machinery is utilized to start, control the speed, and stop the motor automatically, independently of any operator.

A considerable proportion of the large mine hoists now in use have certain automatic features, particularly protective devices against overwinding, and, in some classes of electric hoists, devices for preventing excessive acceleration or retardation. The large automatic hoists discussed in this paper, however, are completely automatic, *i.e.*, capable of making their trips without the presence of an operator at the control levers.

According to circumstances, various advantages may be obtained by automatic control, chief of which are decreased power consumption, increased precision and safety of operation, and decreased cost of attendance. The first step in the analysis of a prospective automatic mine hoist is to determine whether automatic operation is feasible at all. If men are to be hoisted, or levels changed, the attention of an operator is required for these purposes; but under some conditions it may be entirely practicable and advantageous to build the equipment so that, while provision is made for hoisting men or changing levels, ore can be hoisted automatically from any one level. If, however, an operator's attention is required every few minutes for changing levels, handling men or drills, or other work requiring hand control, it is obvious that automatic operation between times will not be of any practical benefit.

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Speed Conditions Required for Automatic Hoisting

For a very slow hoisting speed it may be possible for the skip or cage to pass through the dump at full speed, and a sufficiently accurate stop may possibly be obtained automatically by cutting off power and applying the brakes at full speed. In this case, either a shunt-wound direct-current motor or an induction motor may be used. A number of slow-speed automatic hoists are arranged in this manner and are driven by induction motors. One equipment of this type used in mining work is the inclined hoist for handling concentrates at one of the mills of the Arizona Copper Co., described by H. L. Hall in the *Mining and Engineering World* for Apr. 10, 1915. This hoist has a rope speed of approximately 275 ft. per minute.

For higher rope speeds, at least for speeds over 400 ft. per minute, it is necessary to consider carefully the speed characteristics obtainable from the type of drive proposed. For these higher rope speeds, it is necessary to slow down before entering the dumping horns. Furthermore, the speed about midway in the dump must usually be reduced below the maximum safe speed entering the dump. A reasonably accurate stop is always required; in some cases a total variation of 2 or 3 ft. might not prove prohibitive, but in other cases the stop must be more accurate. For reliable operation, it is nearly always imperative that the automatic-control system shall act in like manner irrespective of load, *i.e.*, that the rate of retardation and the position of stopping be nearly the same whether the skip comes up loaded or empty.

There is only one class of motive power which is inherently suited for automatic operation at high rope speeds, *viz.*, the direct-current shunt-wound motor with voltage control. The speed-torque characteristics for an equipment of this character are represented in Fig. 1. These curves are typical of this class of equipment, although the exact slope of the curves will vary slightly in individual cases. Curve 1 shows the characteristics on the lowest, and curve 5 the characteristic on the highest, speed position of the controller for the case selected. The intermediate curves represent three controller points arbitrarily selected out of a total of 30 or more. It will be observed that these curves are nearly, but not quite, parallel. That is to say, the increase in speed in passing from full load to no load is approximately, but not exactly, the same for the various positions of the controller. The deviation from parallelism is due to the effect of armature reactions in the generator and hoist motor, and may be somewhat different for different cases; but its effect is negligible.

The net advantages (for the purpose of automatic hoisting) obtained by this system of drive are as follows:

As the hoist controller is moved back toward the off position the

hoist is retarded. In case the net rope pull is sufficient and the stored energy of the moving system is not too great, the hoist motor simply drops back in speed to correspond to the reduced generator voltage obtained on the intermediate position of the controller. If, however, the net rope pull is very low (particularly with empty skips in balance), and if the stored energy of the moving system is high, the hoist motor will invert, momentarily, and will act as a generator, returning power to the motor-generator set. This effect is represented in Fig. 1 by the extension of the curves below zero torque. In this manner, if the controller

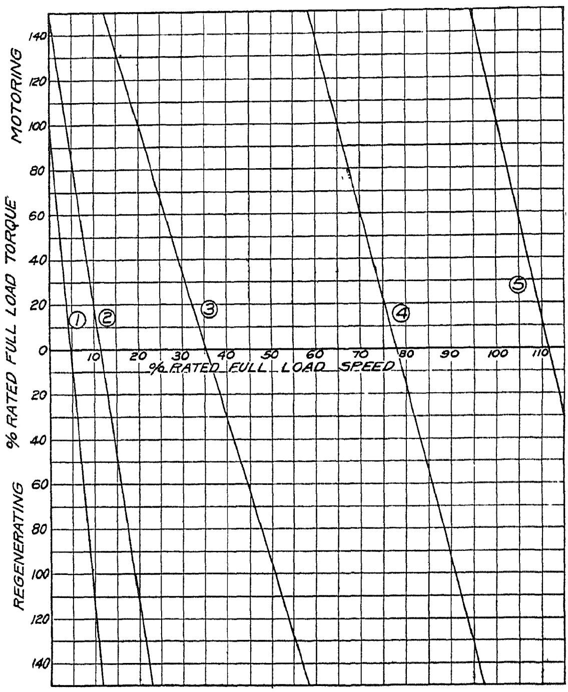


FIG. 1 —TYPICAL SPEED-TORQUE CURVES FOR DIRECT-CURRENT MINE HOIST WITH GENERATOR FIELD CONTROL.

is moved toward the off position more rapidly than the hoist tends to come to rest under the influence of the load, the hoist motor forcibly retards the hoist. If the controller is moved back at the same rate in both cases, the hoist will be retarded to nearly the same speed, and in nearly the same time, irrespective of load in the skip.

It is fairly obvious that the steam hoist is unable to approach very closely the speed conditions just described. The steam hoist, of course, is capable of retarding a load by working against the steam or compression, but the vital points in relation to automatic hoisting are: (1) for the same throttle opening and cutoff, the speed will vary widely with

variation in load; and (2) if the throttle is partly closed or the cutoff advanced to a point at which the skip will enter the dump at a suitable reduced speed, the engine will exert only a slight retarding torque (if any) to help retard from full speed to the reduced speed at which the engine tends to continue. Most of the retardation must therefore come from the load, which is variable, or may even be negative. Furthermore, with a partly closed throttle the final speed, at which the engine tends to continue, will vary widely with variation in load.

The induction-motor hoist, in its relation to automatic hoisting, has somewhat the same characteristics as the steam hoist. Fig. 2 represents the speed-torque characteristics of a typical mine-hoist induction motor. In a direct-current hoist, a given retardation can be accomplished in a certain time and distance by the same manipulation of the control, irrespective of the load hoisted. In a steam or air hoist or an

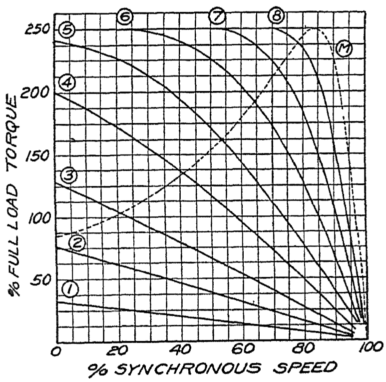


Fig. 2.—Typical Speed-Torque Curves for Induction-Motor-Driven Mine Hoist.

induction-motor hoist, a like retardation of different loads requires different manipulation of the control.

These characteristics indicate, and their further consideration confirms, the conclusion that high-speed mine hoists which are to be operated automatically must be, in almost all cases, driven by direct current.

The Automatic Hoists of the Inspiration Consolidated Copper Co.

When the layout of their main shafts was under consideration by the Inspiration Consolidated Copper Co., a concurrence of several conditions indicated the possibility of effecting a saving by hoisting the ore automatically. These conditions were as follows: (1) A direct-current equipment was necessary in any case, as a motor-generator set was required for the flywheel equalization as provided in the power contract

with the Reclamation Service. (2) The ore was all to be hoisted from one level. (3) Drills, timbers, supplies and waste were to be handled through a drift opening. (4) Men were to be handled on a separate hoist exclusively. (5) On account of the moderate depth and rope speed only a moderate retardation effort would be required.

General Arrangement of Hoists

Two three-compartment shafts have been sunk, for two independent balanced hoists, each hoist being adequate in an emergency to keep the concentrator operating at practically full capacity. The third compartment of one shaft contains a double-deck man cage, operating against

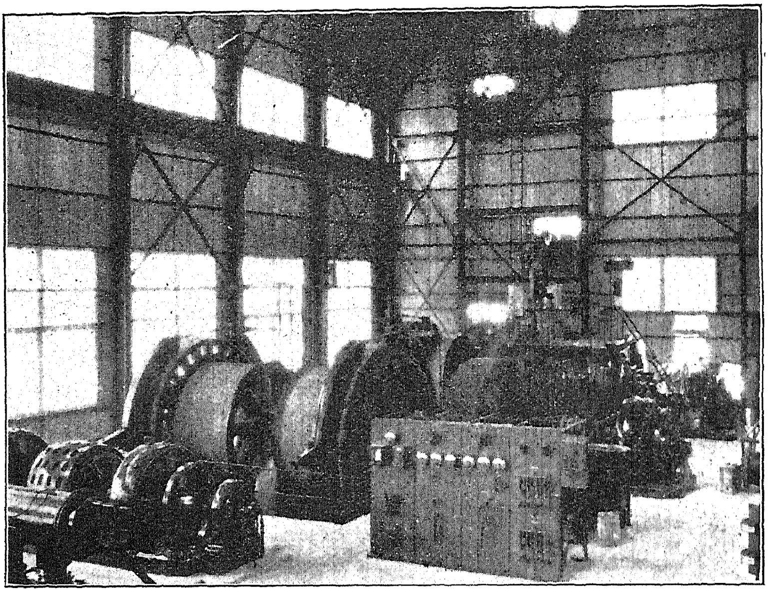


FIG. 3.—MAIN HOISTS, INSPIRATION CONSOLIDATED COPPER CO.

a counterbalance weight, and the third compartment in the other shaft carries this counterweight, together with air lines, electrical cables, etc. Skips carrying 12½ tons are used, and the ordinary hoisting schedule for which the equipment was designed called for an output of 10,000 tons, with a maximum capacity of 14,000 tons, in 14 hr. The hoists are located in one end of the compressor house. No. 2 hoist, in the background in Fig. 3, handles the skips in the East shaft, which is nearest the compressor house. No. 1 hoist, in the foreground, handles the skips in the West shaft, the ropes from No. 1 passing above No. 2 hoist over idler sheaves on the upper deck of the East headframe, thence over the sheaves on the West headframe. Fig. 4 shows the arrangement of shafts and headframes in relation to the compressor house.

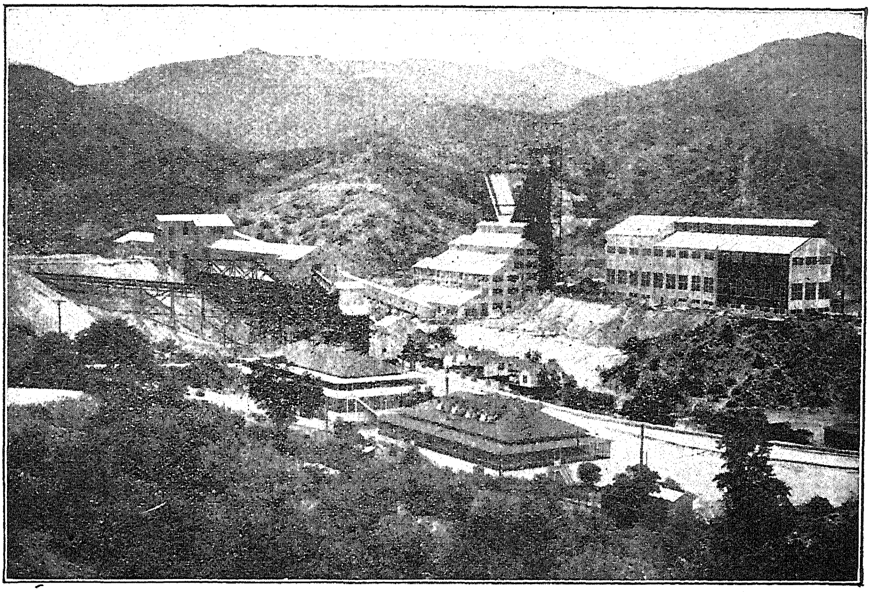


Fig. 4.—Main Shafts, Compressor House, Coarse-Crushing Plant and Storage Bins, Inspiration Consolidated Copper Co., Miami, Ariz., During Construction Period.

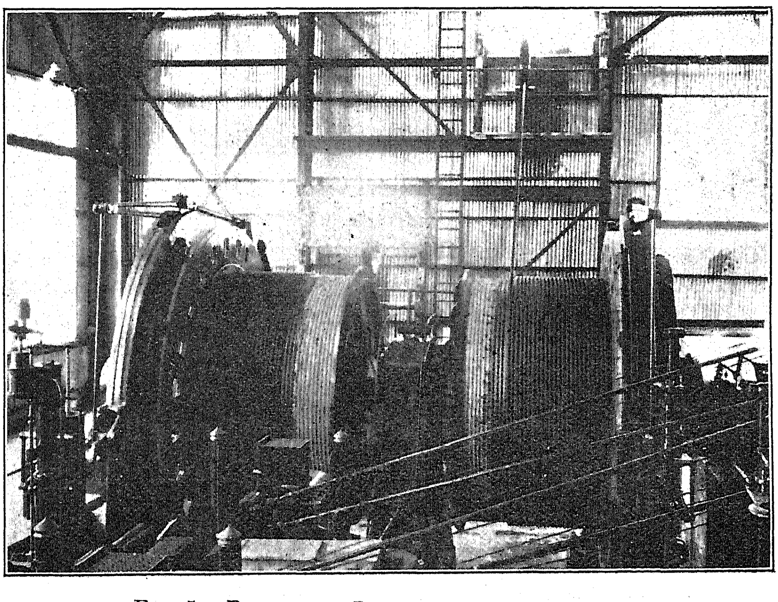


Fig. 5.—Drums and Brakes of One Main Hoist.

The hoists are duplicates, each consisting of one fixed and one clutched drum, each 10 ft. diameter by 65-in. face, grooved for 1,000 ft. of 1¾-in. rope in one layer. The brakes and clutches are air operated with oil cataracts and floating levers, and the automatic control system was so designed that the brake engines could be made practically standard (Fig. 5.) The hoists were designed and built by the Nordberg Manufacturing Co. and the electrical equipment by the General Electric Co.

Each hoist is driven by a 580-hp., 575-volt, 264-r.p.m. shunt-wound motor through a flexible coupling and Falk gears. Power is supplied to the hoists by a 750-r.p.m. flywheel motor-generator set, consisting of one 850-hp., 2,300-volt, 25-cycle induction motor, two 500-kw., 575-

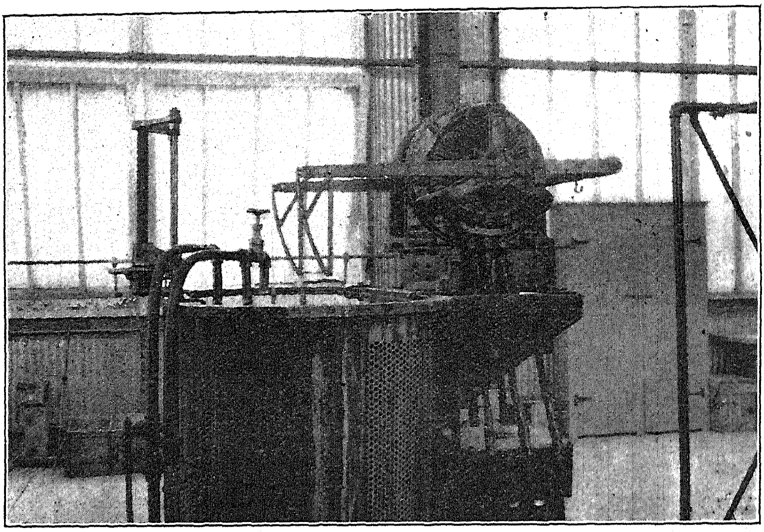


FIG. 6.—LIQUID SLIP REGULATOR FOR REGULATION OF INPUT OF FLYWHEEL SET.

volt generators, one 20-kw., 125-volt exciter and a 19,700-lb. 112-in. diameter steel-plate flywheel. Each hoist motor is connected separately to one of the generators and controlled by varying the field of its generator. The flywheel is not in any way necessary to the control or automatic operation of the hoists. Its function is to eliminate the peak-loads from the power system. The control for equalization of the power demand follows along standard lines, using a liquid slip-regulator for varying the speed of the flywheel set by varying the resistance in the secondary circuit of the induction motor (Fig. 6).

The depth, from the dump to the chairs under the loading pockets, is 630 ft. in each shaft; from the collar to the chairs, 557 ft. The rope speed is approximately 750 ft. per minute.

Description of Automatic Cycle

Before beginning automatic operation it is necessary, of course, that each hoist be properly clutched-in for the loading level, with one skip in each shaft resting on the chairs below its loading chute. It is not important which skips are on the chairs, provided, of course, that the operator obtains a "release" of skips in both shafts before starting the automatic operation. He then introduces the automatic control by closing two small control switches and locking in two levers, all on the operating-platform. This does not, of itself, start the automatic operation, so that the hoists may be left standing in this manner indefinitely. To start the automatic operation, a master controller is thrown to the automatic running position, and left there as long as automatic hoisting continues. According to the positions in which the skips have been resting, one hoist or the other will start. Say, for example, No. 1 hoist starts, hoisting its South skip. The closing of the master controller just mentioned energizes a small pilot motor which moves No. 1 hoist controller gradually to the full-speed position in one direction. As No. 1 controller starts away from the off position, it simultaneously energizes No. 1 generator field and actuates a pilot device which releases the brakes on No. 1 hoist. As the controller moves farther toward the full-speed position, it gradually builds up the generator voltage, thereby accelerating the hoist to full speed.

Toward the end of its trip the travel of No. 1 hoist actuates a pilot motor which moves No. 2 hoist controller gradually to the full-speed position in one direction, thereby accelerating No. 2 hoist in a similar manner, to hoist its North skip. Shortly before its skip enters the dumping horns, the travel of No. 1 hoist, by means of cams, one of which is geared to each drum, moves No. 1 controller gradually toward the off position. This gradually decreases No. 1 generator voltage, thereby retarding No. 1 hoist, and just as its North skip is about to land on the chairs, No. 1 controller comes into the off position. This completes the retardation and automatically applies the brakes. No. 1 hoist stands at rest while No. 2 is hoisting its North skip. Toward the end of its trip, No. 2 hoist energizes the pilot motor for No. 1 controller so as to start No. 1 hoist in the opposite direction, *i.e.*, to hoist its North skip. No. 2 hoist comes to rest in the manner described for No.1, and rests while No. 1 is hoisting its North skip. Toward the end of its trip, No. 1 hoist energizes the pilot control to start No. 2 in the opposite direction, *i.e.*, to hoist its South skip. The sequence continues in this manner until stopped by the operator, as described later.

A loading system is used underground by which the skips are automatically loaded with a predetermined weight of ore per trip. The reduction of the attendance required at the foot of the shaft contributes

materially to the advantages of automatic hoisting. The automatic loading system can be thrown out of engagement in either shaft so that the hoists may be operated either automatically or by hand, for purposes of inspection or adjustment, without hoisting any ore.

Variation in Rate of Automatic Hoisting

To obtain a more rapid operation of the hoists, *i.e.*, a greater number of trips per hour, when operating automatically a control switch may be thrown, by which each hoist will be started earlier in the trip of the other hoist, thus overlapping to a greater extent the trips of the two. If it is desired to run the hoists automatically at fewer trips per hour than normal, this is done by introducing resistance permanently in each generator field circuit, to give a rope speed lower than normal.

Hand Control

When the details of design were first considered, one of the chief problems was the arrangement of the control so that the transition from hand to automatic operation, and more especially the transition from automatic to hand operation, might be made without risk or delay, and in a manner easily remembered by any operator acquainted with the equipment. To this end the levers on the operating platform which operate the hoist controllers and brakes for hand control are not disconnected from the controllers or brake engines when running automatically. Consequently, when the automatic pilot devices are cut in, and the hoists are operating automatically, these levers move back and forth, as if the hoists were being controlled by hand by invisible operators. When, therefore, the transition from automatic to hand operation is made during a trip, the brake and controller levers of both hoists are in the correct positions and properly in engagement for hand control.

The automatic operation can be interrupted at any time during a trip. This is done most easily by throwing the master controller for automatic operation to the off position, which causes any trip which is under way at the time to be completed automatically, dumping in the usual manner, but prevents the next trip from starting. If the hoists are then left standing, and not operated by hand, all that is necessary to start automatic hoisting again is to throw the master controller to the automatic running position (Fig. 7).

Before the construction work at the foot of the shafts and in the bins in the tipple had been completed in all details, it was necessary occasionally to stop an automatic trip without letting it dump. In such an event, or when necessary for any reason to transfer to hand control before completing a trip, the master controller for automatic hoisting is thrown

to the off position. Without disconnecting or unhooking any other parts the controller lever of the hoist which is running may then be pulled back to the off position by hand, and as the controller comes into the off position the brakes will set automatically. It is now possible to leave the pilot control of the brakes connected in service, so that the brakes will release and set automatically, as the controller is moved by hand from or to the off position. Or, if necessary on account of the character of hoisting to be done, the automatic pilot control of the brakes can be cut out; in which case brakes and controller will be controlled separately by hand.

Under all conditions (except when making adjustments in the manner

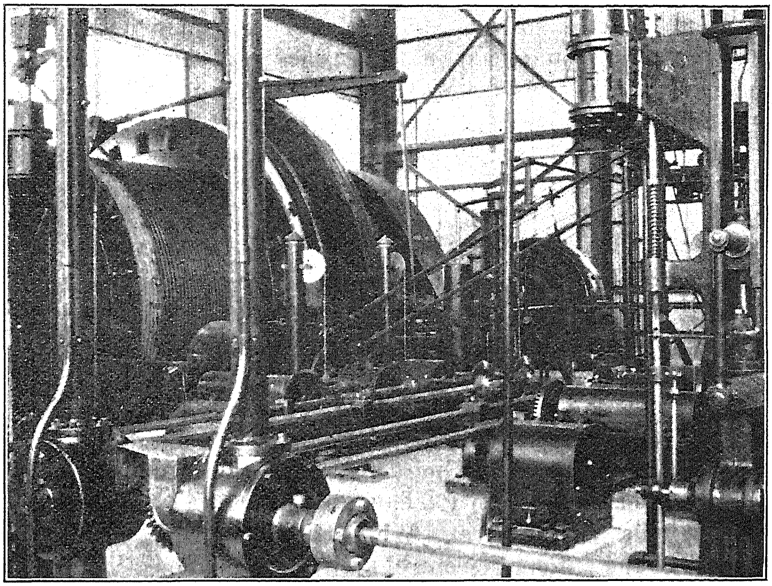


FIG 7.—DEPTH INDICATORS AND AUTOMATIC-CONTROL SYSTEM FOR MAIN HOISTS.

described later), the cams on each hoist controller remain connected mechanically to the hoist drums. This cam mechanism thus serves two purposes: (1) in automatic operation it provides the automatic slow-down and stop; and (2) in hand operation, if the operator does not begin retardation at the proper point, this mechanism will retard the hoist in practically the same manner as when hoisting automatically, thus providing protection against overwinding when operating by hand.

Protective Devices

The protective system resembles those of a considerable number of large direct-current mine hoists, of the same general type (except the

automatic operation) as the Inspiration hoists. In the latter, as has just been noted, the automatic control system provides against overwinding in hand operation. An additional set of emergency-limit switches is used, which gives similar protection in case of failure of the automatic control. During hand operation there are effective, therefore, two complete sets of protective devices against overwinding. For each hoist a hand-operated emergency switch is provided on the operating platform, and a similar emergency switch is located at the foot of the corresponding shaft, by means of which either or both hoists may be stopped quickly from the operating platform, or the foot. Without appreciable complication, additional emergency switches may be installed at other points, if desired.

The operation of any one or more of these emergency devices cuts off power from the hoist and makes an emergency application of the brakes. An emergency, which affects one hoist only, acts on the power and brakes of that hoist only. The failure of excitation or alternating-current power, which affects both hoists, cuts off power and makes an emergency application of the brakes simultaneously on both hoists.

Adjustments in Service

When unclutching for changing levels, and when taking up stretch of ropes, the adjustments are taken care of as follows:

On the Inspiration hoists it has been the custom, whenever the hoists are to be idle an entire shift, to bring both skips to the collar of the shaft, in order to save rusting of the ropes. This is done by unclutching just as in any ordinary hoist with one fixed and one clutched drum. If desirable for any reason, either hoist may be run by hand control either out of balance or clutched in for balance to operate from other levels than the regular loading level. When clutching or unclutching, the adjustments of the automatic control system are not touched.

If the shafts are sunk to the ultimate depth contemplated, and the present loading stations abandoned, the control can be readjusted to operate automatically from the increased depth. Without changing the adjustment of the control equipment, it is not possible to operate automatically from levels differing considerably from the normal level for which adjustment has been made, but the system is capable of modification so as to hoist automatically in balance from any level to the dump, without readjustment, all the adjustments being taken care of automatically by clutching in at the desired level.

Stretch of ropes is taken up in a simple manner which itself is semiautomatic and does not require any measurements. The first time it was necessary, the stretch was taken up on both ropes of one hoist in about 15 min., at the end of which all adjustments were in shape for hand or automatic operation. The method is as follows:

The hoist is run into an automatic stop with the skip on the clutched side resting on the chairs. (This is effected by the cam which is geared to the clutched drum.) The controller and cams are now in the proper position for an automatic stop on this side; but the rope on this side has unwound farther than normal by an amount equal to the stretch or slack which it is intended to take up. This cam is now uncoupled, but the other cam is left coupled. The hoist is now moved by hand control just far enough to wind up the estimated amount of slack, and the cam on this side is then coupled up to the clutched drum. This operation takes up the slack on the clutched side and transfers it to the fixed side. The hoist is now run, in balance, into an automatic stop on the fixed drum side, which lands the skip on the fixed drum side on the chairs, and brings the skip on the clutched side into the dump. The cam on the fixed side is now uncoupled, and before moving the hoist to take up slack, the other drum is unclutched, so as to leave its skip in the normal position in the dump. The fixed drum is then moved sufficiently to take up all the slack on that side, *i.e.*, the stretch of rope on that side plus the slack transferred to that side by taking up the stretch on the clutched drum side just previously The cam is then coupled up to the fixed drum and the other drum is clutched-in, which completes the adjustment of both ropes and cams and leaves the hoist ready for operation. It is necessary, of course, not only to take up stretch on each side, but also to clutch-in at the proper level. During the foregoing procedure, after unclutching one drum as described, the same movement of the other drum which takes up the slack also makes the necessary correction for level.

General Observations on Operation

The East shaft was ready for operation before the construction work had been completed in the West shaft. The ropes were put on No. 2 hoist, and for purposes of test and for a thorough tryout of the system, both hoists were operated automatically, No. 1 hoist running automatically as if in actual service, but without any ropes on the drums.

Both ropes were on No. 2 hoist and both skips were hung in the East shaft by the morning of July 25, 1915. During one shift on that day, after marking the ropes and the drum flanges, we coupled up the automatic control and the depth indicators to the drums, checked up the shaft and tipple clearances, and the adjustment of the cams for automatic retardation and stop, and hoisted 18 skips of ore by hand control, using the cams for automatic retardation but not using complete automatic operation. The following day, between 8 a.m. and noon, we made adjustments for complete automatic operation, and hoisted automatically 44 loaded skips. The adjustments were refined somewhat at a later date, but those made during the first three-quarters of an hour of automatic operation worked well.

The same morning in which the equipment first operated automatically, the accuracy of stop was observed for 12 consecutive trips, *i.e.*, six trips each way. The total variation between maximum and minimum was 4 in. of rope travel. After a few weeks of intermittent operation, similar observations were taken. In 20 consecutive trips (10 each way), the total variation between maximum and minimum was only 1.5 in. of rope travel in one direction and 1.25 in. in the other. During this time the ore hung back in the loading pockets on one side, so that six of the trips included in the above figures were made empty. It is significant that this variation of 1.5 in. is only 1 per cent. of the distance traveled per second at full speed of the hoist.

Attendance

To operate two hand-controlled hoists, either steam or electric, of the size and importance of these, would require at least two operators per shift; and according to practice in some localities, an oiler would be employed in addition to the two operators.

For the operation of these two automatic hoists there is required only one operator, who is able to attend to the oiling and to whatever hand operation of either hoist may be necessary on his shift.

General Conclusions

In the section of the paper dealing particularly with the Inspiration hoists, it is our purpose principally to describe the operating features and the results accomplished by the system, as it would expand this paper to an excessive length to describe fully the essential details of design of the control equipment by which these results were attained. It would enlarge the paper still more to enter into the reasons in accordance with which the methods were selected for performing the several necessary functions of this control system. When the design of the equipment was undertaken several engineers worked, at first independently and then in consultation, in order to give due consideration to all practicable methods of operation of the various details, and as a result several arrangements were studied and discarded before settling on the one finally adopted. It was this thorough preliminary study of the entire situation which made it possible to begin practical automatic operation promptly, after the skips were hung in the shaft.

Differences in operating conditions will naturally require different methods of control, so that it is to be expected that for large automatic mine hoists which may be built in the future the control system will differ from the Inspiration system in several respects. However, the exhaustive investigation of the subject which preceded this installation

and the experience gained in the adjustment and starting of the Inspiration equipment, make it possible to determine readily the feasibility of other proposed automatic hoists for different conditions of operation. The experience thus gained makes it practicable, moreover, to build equipments for greater depths, higher speeds, or other exacting conditions for which, without this experience, it would be impossible to make designs with reasonable certainty of success.

The application of automatic mine hoists will always be limited by the fact that operation cannot be truly automatic, except where the conditions of hoisting are reasonably uniform. In other words, where, under prevailing conditions, the attendance of an operator is required practically continuously throughout the shift in order to change levels, hoist or lower loads out of balance, or handle men, it is impossible to realize any practical advantages by operating automatically during the short periods of hoisting ore regularly from any one level. On the other hand, entire uniformity is not necessary in order to make automatic operation practicable. As an illustration, consider the case of a main hoist serving a few levels, and an auxiliary hoist in the same hoist house handling all men, timbers, supplies, waste, etc., for all the levels served by the main hoist. Conditions of operation may possibly be sufficiently favorable so that if the main hoist is arranged for automatic operation (or for semi-automatic control from the level stations by the skip tender), the operator for the auxiliary hoist will be able to take care of the hand operation required on the main hoist.

It may reasonably be anticipated that from time to time various mine-hoisting projects will come up for consideration in which the possibilities offered by automatic hoisting should by no means be dismissed without investigation.

Comparative Friction Test of Two Types of Coal Mine Cars

BY P. B. LIEBERMANN,* NEWARK, N. J.

(Arizona Meeting, September, 1916)

THE resistance to motion offered by mine cars is caused principally by: Rolling friction, flange friction, bending rails, bearing friction and wind resistance. With proper construction and with a fair amount of attention and care, that is, under favorable conditions, the first three items cannot be greatly improved. The wind resistance ordinarily is negligible on account of the slow speeds commonly used. The bearing friction, however, is a variable item, depending upon the design, the bearing material, the lubrication, the location (whether in wheel-hub or on car-body) and the alignment. Under the most favorable conditions the bearing friction will reach a lowest possible value. By the development of a successful anti-friction mine-car bearing this lowest value has been improved to such an extent as to affect the whole system of mine haulage.

After being satisfied of their durability and certainty of operation, any investigation into the advantages of anti-friction bearings for mine cars will have to be based upon reliable information as to the saving in power of such bearings. While numerous tests have been made in this direction, the results have not always been convincing, principally because of the rather crude methods that were employed. These crude methods are the main cause of the wide variations existing in regard to the exact amount of the frictional resistance.

Frequently an ordinary spring balance is used and the pull necessary to move a certain car or train of cars is obtained by a number of readings. Such spring balances necessarily have a coarse graduation and the continuous vibration and fluctuations to which they are subjected make them highly dependent upon the individual observer. Further, they must be frequently calibrated. Sometimes these tests were made without regard to acceleration, speed or even grade.

The method of using an electric locomotive as a testing device is perhaps more open to criticism than the spring-balance method, because it introduces additional sources of error: The varying electrical and mechanical losses in the locomotive. Also, such a measurement always includes the power necessary to move the locomotive itself.

This means a considerable amount of figuring, and the results due to the various multiplications and corrections may be of little value, for obvious reasons. When figuring the drawbar pull from the electrical

* Engineer of Tests, Hyatt Roller Bearing Co.

readings, a proper connection between drawbar pull, speed and location on the track is not easily established on account of the separate observations of the individual operators. Additional errors can be caused by changing locomotives during a series of tests, by the use of sand during a test or by operating the controller on resistance points and making no allowance for this.

When it was decided some time ago to start a thorough investigation into the merits of flexible roller bearings in comparison with ordinary plain bearings for mine-car use, it was clearly seen that it would be very desirable to design a special testing device for that purpose. The following requirements suggested themselves as desirable in making the testing device convincing and beyond criticism:

1. Take all readings graphically, that is, by means of recording instruments, so as to be free from the influence of the individual operator. Arrange all readings in their proper relation and side by side on a continuous strip of properly graduated record paper.

2. Eliminate corrections and reduce the amount of figuring to a minimum.

3. Exclude the locomotive and measure train resistance, or drawbar pull, directly.

4. Determine the speed of the train for every drawbar pull reading.

5. Determine the time taken to travel over a certain section of the road.

6. Determine the corresponding position on the track of the train under test and show the grade on which the train is running.

A testing device combining all these requirements has been designed by the author in the form of a dynamometer car (Fig. 1). This car is absolutely self-contained and is intended to be placed in front of the cars that are to be tested and behind the locomotive.

The dynamometer car contains the following instruments:

1. A hydrostatic pressure diaphragm, connected by means of a heavy brass tube to a pressure gage and recording pen. The whole system is filled with oil and is permanently sealed after great care has been taken to expel every particle of air.

2. An electric speedometer. This consists of a good-sized magneto, chain-driven from one of the axles, and a recording voltmeter, calibrated to register directly miles per hour.

3. A time-marker clock, which makes a mark on the record paper once every 5 sec. and a double mark every minute.

4. The apparatus for driving the record paper. The paper travels from the paper roll over a table and past the recording pens to a drum which is chain-driven from one of the axles.

The three readings: Drawbar pull, speed, and time, are recorded simultaneously and are so arranged as to be always in synchronism.

The profile of the road over which a test is run is added by hand on

a wide margin specially provided for that purpose on the record paper, in its proper relation to the graphical records.

As the paper is driven from the axle the paper-travel is a function of the distance covered by the train. For every mile the train runs, the paper travels 33 in. Heavy divisions on the record paper represent 0.1 mile; fine divisions represent 0.01 mile. Under the conditions the records cannot be integrated for mean values, except for periods during which the speed is fairly constant.

In the design of this dynamometer car, the example of the main railroads was followed and their experience with similar outfits was utilized

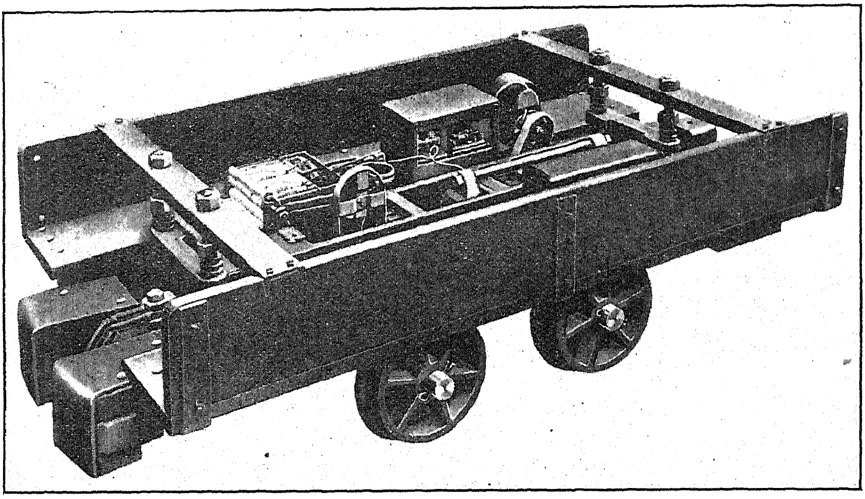


FIG. 1.—DYNAMOMETER CAR.

to a large extent. But while in railroad practice conditions are pretty well standardized, this is not the case in the different mines. Here the size of the haulageway limits the overall dimensions of the dynamometer car; hence, the car must be so proportioned as to go into the smallest shaft without interference with the roof or the walls. In addition, there exists a large variety of track gages, bumpers and couplings. Further, there is no standard for the distance from the top of the rails to the center of the drawbar. To make the dynamometer car universal it was necessary to make it take care of the different local conditions. To run on any gage of track it was provided with different lengths of axles which could easily be interchanged. The axles were arranged to hold the wheels in certain places for accommodating track gages from 30 in. to 48 in. The varying height of drawbar was taken care of by arranging the dynamometer with all the recording instruments on a large cast-iron frame which could be raised or lowered on four posts to suit conditions. The bumpers were so arranged and proportioned as to operate in conjunction with

either side bumper cars or center bumper cars. The drawheads were arranged to accommodate any existing coupling.

The dynamometer car was built according to the author's specifications and under his supervision at the shop of the Tinius Olsen Co., Philadelphia.

A number of tests have been made with the dynamometer car since it was put into commission late last summer. The test described in this paper, while the least favorable for the roller bearings, is of considerable interest due to the fact that it was witnessed by a large number of mining engineers and other critics.

Right at this point it should be remembered that as far as the running gear of mine cars is concerned there are wheel-hub bearings, bearings in outside boxes, and bearings in inside boxes depending on the location of the bearings in relation to the wheels. Besides this there are axles with two loose wheels, axles with one tight and one loose wheel, axles with two tight wheels and axles split in the middle with one wheel on each half. These different arrangements of bearings and axles will undoubtedly affect the train resistance and while a split axle with bearings in inside boxes like the Anaconda type represents the best engineering practice, it is obvious that for a particular investigation only such an arrangement can be considered as is representative of the conditions existing in the respective territory. In the bituminous-coal mines wheel-hub bearings are used almost exclusively and therefore a comparative test conducted in a bituminous mine should be based on bearings located in the wheel hubs. Such a test is described in this paper.

This particular test was made at the Greensburg Coal Co.'s mine at Greensburg, Pa., in coöperation with the H. C. Frick Coke Co., which was represented by its Chief Mechanical Engineer, C. E. Huttelmaier. The test was run on the main haulage road underground to determine the saving in tractive effort of mine cars equipped with roller bearings as compared with mine cars equipped with plain bearings under identical conditions of operation.

The following types of mine cars were tested:

Item No.	Type of Wheel	Location of Bearing	Diameter of Axle in Bearing, Inches	Length of Bearing	Track Gage, Inches	Diameter of Wheels, Inches	Wheel Base, Inches	Drawbar to Top of Rails, Inches
1.	Roller bearing	In wheel hub	2⅜	8 in. length of rollers	40	18	27	15½
2.	Plain bore	In wheel hub	2⅜	8 in. length of hub	40	18	27	15½

These are steel cars built by the Hockensmith Wheel & Mine Car Co., and are identical in every respect except bearings. The roller bearings have a planished outer race with the flexible rollers running directly on

the axle. The plain bearings are formed by the hub of the cast-iron wheels (Fig. 2). The end thrust in each case is taken care of by the wheel hub rubbing against the cast-iron axle box on the bottom of the car. The cars equipped with roller bearings had been in service about two years before this test was made. As there are no plain-bearing cars in the Greensburg mine, the wheels of 20 roller-bearing cars were removed and replaced with plain bore wheels. The plain bearings were run for some three weeks before this test was made; this was considered a sufficient length of time to wear these bearings down to a good bearing seat. As the hub of the plain bore wheels was somewhat shorter than the hub of the roller-bearing wheels, a washer of proper thickness was inserted at each end of the plain bore wheels. All roller bearings were filled with fresh grease before the test. The old grease was not drawn off nor was

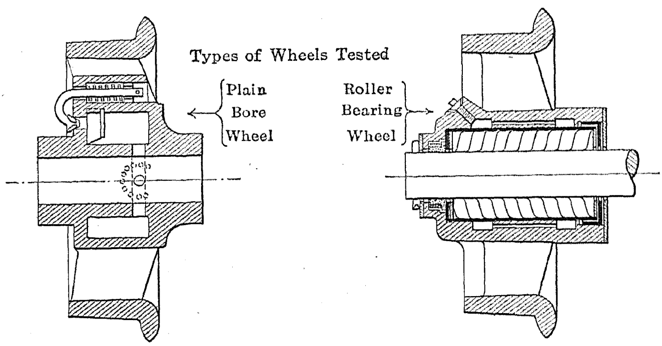


FIG. 2.—TYPES OF WHEELS TESTED.

any cleaning or overhauling done. The lubricant used was a thin grease. The plain bearings were lubricated with ordinary black oil prior to this test.

For the actual test two different trains of 20 cars each were used, one train being made up of cars with roller-bearing wheels, the other train consisting of cars with plain bore wheels. The cars of the two trains were loaded as uniformly as possible with coal, just as obtained in every-day practice. The brake action was the same on all cars as nearly as could be judged. Each car weighs loaded 7,150 lb.; therefore each train of 20 cars weighs 143,000 lb. or 71.5 tons. One train at a time was tested by pulling it by means of an electric locomotive over the 4,116 ft. of track whose profile is shown on the charts; the dynamometer car was inserted between the locomotive and the train in such a way that its own resistance is not indicated on the record charts. The test of each type of train was repeated three times.

The rails of the track weigh 40 lb. per yard. They were dry and in good condition when these tests were made. A careful calibration of the dynamometer car before and after the test showed all instruments indicating correctly.

The results of the tests are recorded on the charts, two copies of which are shown in Figs. 4 and 5. Instantaneous values of drawbar pull and speed for any place on the track can be seen on the charts at a glance. For average values a piece of tangent track of sufficient length and of nearly uniform grade and over which the speed was fairly uniform was selected. The average drawbar pull and speed were obtained from the charts by the use of a planimeter.

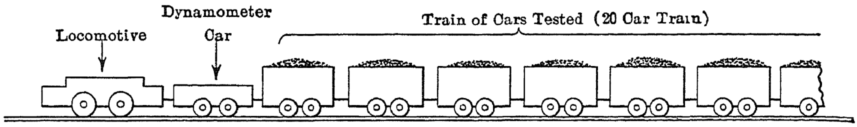


FIG. 3.—TEST TRAIN.

The following average values were found:

Type of Bearings	Average Grade, Per Cent.	Test No	Average Drawbar Pull, Pounds	Average Speed, Miles per Hour
		4	4,410	5.75
Roller	2 4	5	4,350	6.56
		6	4,280	5.56
		1	5,250	5.70
Plain	2.4	2	5,070	5.78
		3	5,190	5.30
Average of tests 4, 5 and 6			4,347	5.98
Average of tests 1, 2 and 3			5,170	5.59

To correct these figures for grade, 20 lb. per ton per 1 per cent. must be subtracted from the above drawbar pull. For the train weight of 71.5 tons and the average grade of 2.4 per cent. the correction amounts to 3,432 lb. for each train.

The corrected tractive effort for the roller-bearing cars therefore amounts to:

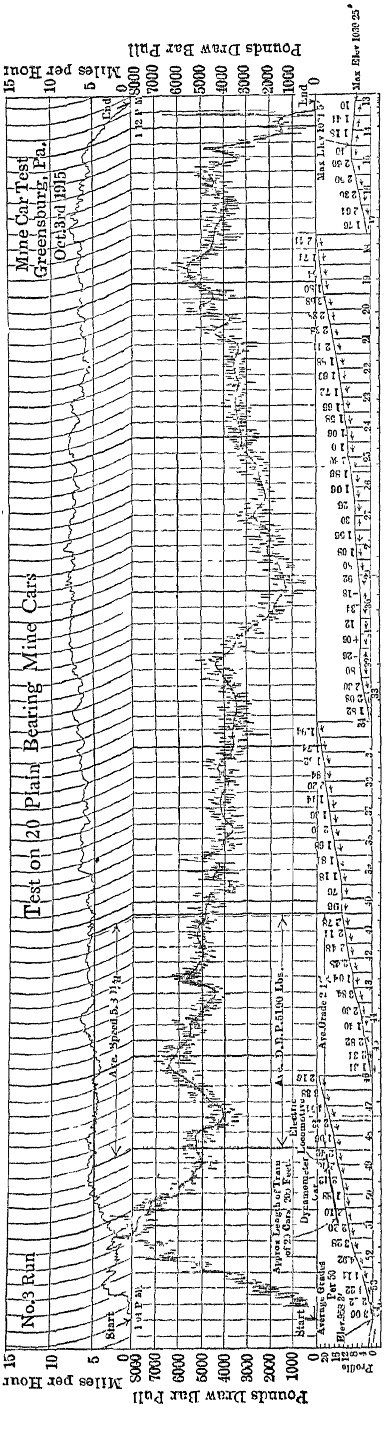
$$4{,}347 - 3{,}432 = 915 \text{ lb. at } 5.98 \text{ miles per hour}$$

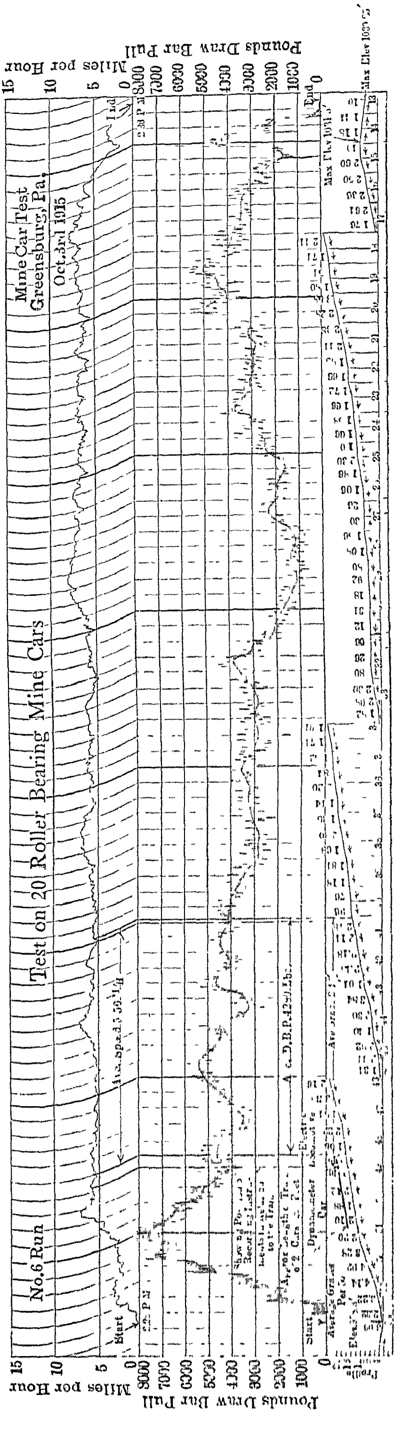
and for the plain-bearing cars:

$$5{,}170 - 3{,}432 = 1{,}738 \text{ lb. at } 5.59 \text{ miles per hour}$$

Expressed in pounds per ton this means for the roller-bearing cars 12.8 lb. per ton and for the plain-bearing cars 24.3 lb. per ton.

Disregarding the slight difference in speed, the saving in drawbar pull in favor of roller-bearing cars therefore amounts to 47.25 per cent. In other words, a locomotive of a certain size would be able to pull the following number of cars provided the speed is the same in each case:





FIGS. 4 AND 5.—CHARTS OF TEST RUNS.

Grade, Per Cent	Per Cent. More Roller-Bearing than Plain-Bearing Cars
Level	90
1	35
2	22
3	16

Although considerably greater differences between the flexible roller bearings and the plain bearings have been obtained by tests with the dynamometer car in other mines, the above figures mean reductions in operating expenses or an increase in production large enough to command attention. Inasmuch as the coefficient of friction of different types of rollers of the same dimensions will not vary over wide limits, the present investigation clearly demonstrates the possibilities of a well-built anti-friction bearing compared with a plain bearing.

Besides the motor haulage (the size of the locomotives, transmission line and power house) one important additional point should not be lost sight of, the gathering of the cars by hand. Actual experience has shown that a certain size car when equipped with plain bore wheels needs three men for starting and running while the same car when equipped with flexible roller bearings needs only one man for starting and running While no starting tests have been made during the above described tests at Greensburg, sufficient information has been obtained at another mine to show the starting pull of plain-bearing cars to be about 150 per cent. higher than for flexible roller-bearing cars. It was found that starting tests require a great amount of care to obtain reliable results and such tests will be the special subject of a later investigation.

It will be noticed that for all running tests a piece of track with a pronounced grade was selected. This was done purposely as it was found that the effect of the pull on the instruments was far more uniform and steady on a grade than on the level; this, in turn, means that the results after being corrected for grade are more certain. To be sure, a number of tests have been run on level track and the results obtained check in every instance with the corrected results of tests made on a grade.

In the beginning, the dynamometer car naturally developed some imperfections and great pains were taken to weed these out. Improvements have repeatedly been made.

It is intended to keep the dynamometer car in commission until the various conditions existing in the different coal and metal mines all over the country have been thoroughly investigated. An analysis of these various tests will be the subject of a later paper. At that time it is expected that sufficient material will be available to form definite ideas about the relation between drawbar pull and speed, the effect of car weight, the effect of different track gages, of curves, of different weights of rails, of different wheel diameters, of different lengths of trains, the difference between empty and loaded trains, the effect of weather, etc

Discussion

Edwin M. Chance, Wilkes-Barre, Pa. (communication to the Secretary*).—I have read with great interest Mr. Liebermann's paper reporting dynamometer tests of mine cars and wish to express my appreciation of this investigation. The coal-mining industry needs just such precise study of the problems with which it is confronted.

It would seem, however, that the writer has demonstrated that a high-grade grease is superior to black oil as a mine-car lubricant rather than that roller-bearings are superior to plain. While it is reasonable to believe that roller bearings will cause less friction than plain when used on the trucks of mine cars, still this point can not be conceded on the strength of the tests quoted, for in these tests grease was the lubricant of the roller bearings, while the usual black oil, of low lubricating value, was the lubricant of the plain bearings. Under these conditions, a case is made for the lubricant rather than for the bearings.

Some time ago certain of the coal mining companies with which I am associated called upon me to have a high-grade grease prepared for them for use in plain mine-car bearings. This problem was successfully solved and this lubricant has now been in use for some years. The fact has been established that the ordinary plain mine-car journals are especially susceptible to such a lubricant and that its use greatly decreases the drawbar pull of these cars. The fact must be borne in mind, however, that the exigencies of this service require a special grease and that the use of the ordinary grades of cup grease will surely spell disaster.

It would be indeed interesting to have tests of the same degree of precision as those reported made upon cars with plain bearings lubricated with black oil and special mine-car grease and to compare the latter with tests under like conditions made upon cars equipped with roller bearings.

Charles Legrand, Douglas, Ariz. (communication to the Secretary†).—Mr. Liebermann's paper on friction tests of two types of coal-mine cars is very interesting. It is a pity that the dynamometer car was not designed for use on tracks of 18 to 24 in., as this is the most usual gage in metal mines in this part of the country.

I have tried to make tests by means of an ordinary spring dynamometer and find, as stated by Mr. Liebermann, that this method is very unsatisfactory, as the conditions of mine tracks are too variable to produce a steady pull on the dynamometer. It would be very valuable to the mining industry to have accurate tests taken on narrow-gage track, to determine not only bearing friction, but the total track resistance. Such tests would convince the mine managers that it pays to lay heavy tracks and keep them in good condition.

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* Received June 24, 1916.

† Received July 5, 1916.

Personally, I have always been in favor of the roller bearing, not only on account of its lower friction but also on account of its smaller maintenance and lubricating costs. To be successful, however, this type of bearing has to be dustproof. On a narrow-gage car this is usually done by inclosing the whole axle in a casting which also supports the two bearings. This casting acts as an oil reservoir and permits the car to be run a whole month with one oiling.

I hope that Mr. Liebermann will continue his work and make tests of the narrow-gage cars.

JOHN PHILLIPS, Pittsburgh, Pa. (communication to the Secretary*).—I am familiar with the test made by the Greensburg Coal Co. and described in Mr. Liebermann's interesting paper, although I was not personally present. Mr. Liebermann's results on the Hockensmith plain-bearing wheels compare closely with tests which have been made elsewhere and show a considerable saving in favor of the roller bearings. However, as the grade increases, the ratio in favor of the roller bearings decreases until, on a 4 per cent. grade, their efficiency is only 11 per cent. greater than that of the plain bearings, as shown by the following official figures of the Hyatt company:

Test, Oct. 3, 1915

	Pounds
Weight of car	2,650
Weight of coal	4,500
Total	7,150

Weight of 20 cars loaded, 2.4 per cent. grade average	71.5 tons.
Correction	48 lb. per ton.

$71.5 \times 48 = 3{,}432$ lb.

4,280	5,190
4,350	5,070
4,410	5,250
3)13,040	3)15,510
4,347 Hyatt	5,170 plain
4,347	5,170
3,432	3,432
915	1,738

Frictional resistance for Hyatt, $\frac{915}{71.5} = 12.8$ lb. per ton.

Frictional resistance for plain bearings, $\frac{1{,}738}{71.5} = 24.3$ lb. per ton.

It will be noticed that all the readings were taken on a straight track on an average grade of 2.4 per cent. Had any of the readings been taken on a curve, we are confident that the drawbar pull on both the roller bearings and the plain bearings would have been considerably higher

* Received July 10, 1916.

Drawbar, Pounds per Ton		Grade, Per Cent	Per Cent. Saving	Percentage Cars Possible to Add in Case of Roller Bearings
Roller Bearings	Plain Bearings			
12.8	24.3	level	47⅓	90.0
32.8	44.3	1	26.0	35.0
52.8	64.3	2	17.9	21.8
72.8	84.3	3	13.6	15.8
92.8	124 3	4	11.0	12.4

when the train was rounding the curve, because the axles of the Hockensmith trucks are of straight cold-rolled steel without collars, and in rounding the curve the car naturally shifts against the inside wheel. The rear hubs of the wheels are necessarily large in order to admit the bearings, and the large hub grinding against the box creates considerable friction, which would increase the drawbar pull on a curve.

Records of three tests conducted by the Engineering Department of the Pittsburgh Coal Co., on three different types of wheels, are given below. At Essen No. 3 mine, the Jarvis Adams roller bearings were used; at the Champion mine, Eureka wheels and angle-bar trucks (the wheels being similar to the plain-bearing wheels used in the Greensburg test), and at the Dickson mine, our open-cap wheel trucks. Inasmuch as the drawbar pull on the Hockensmith plain-bearing wheels at Champion agrees very closely with the results on the Hockensmith plain bearings at Greensburg, the results may be taken as approximately correct, and show the following comparison:

Hyatt roller bearings at Greensburg..................	12.8 lb. per ton.
Jarvis Adams roller bearings at Essen No. 3	15.14 lb. per ton.
Open-cap trucks, Dickson mine	18.48 lb. per ton.
Hockensmith plain bearings, Champion mine...........	24.40 lb. per ton.
Hockensmith plain bearings, Greensburg...............	24.30 lb. per ton.

The figures on the Hyatt bearings were lower than we anticipated, as we had always supposed that the Jarvis roller bearings when new and in good condition were the easiest running on the road. Our open-cap truck costs about the same as the Hockensmith angle-bar truck, but runs about 25 per cent. easier. The cost of the roller-bearing truck is about $10 to $12 per car more than our truck, and on the basis of the Greensburg figures runs about 30 per cent. easier.

From the above data, I think that it is safe to say that it would be well for Mr. Liebermann, or some person equally well qualified to carry out this class of tests, to pursue it much further, both as to different kinds of wheels, and to the character of roads, which would include different grades and varied curvature, before a thorough comparison could be made and a conclusion arrived at that would be entirely satisfactory.

The Water Problem at the Old Dominion Mine

BY P. G. BECKETT,* GLOBE, ARIZ.

(Arizona Meeting, September, 1916)

THE problem of handling the large quantities of water encountered in the Old Dominion mine presents many features of interest. In the present paper are discussed the probable sources of water, the pumping equipment, the increased flow into the mine in the early part of 1915, culminating in the flooding of the 18th and 16th levels in March of that year, and the difficulties that were met with during the high-water period.

I. PRELIMINARY

The Old Dominion mine, situated 1 mile north of the town of Globe, Ariz., has long been one of the prominent copper producers of the State, and until the opening up of the copper schist deposits in the neighboring mines of Miami, was responsible for the greater part of the copper output from the Globe district.

As the gradual exhaustion of its orebodies near the surface led to the downward development of the mine, more water was encountered in the western portions directly underlying the dacite formation until the drainage problem became a serious factor in the operation of the mine and in the economical extraction of the ore.

A heavy inrush of water on the 10th level of the mine in 1906, preceded by still earlier water difficulties, rendered imperative the immediate installation of a pumping equipment capable of handling large flows of water. While the Old Dominion has been recognized as a wet mine since its early days and one in which the complete control of the water situation was of paramount importance in the development of the lower levels, it is chiefly in the last nine years, since the installation of the increased pumping equipment, that the mine has developed and handled large quantities of water.

Its maximum pumping record was reached in the first six months of 1915, during which time, following a prolonged period of abnormal precipitation, the pumps handled 1,624,740,000 gal. of water; while in the month of March alone, over 407,000,000 gal. of water were pumped from the mine.

* General Manager, Old Dominion Copper Mining & Smelting Co.

II. PROBABLE SOURCES OF WATER

For a proper understanding of the fundamental conditions affecting the flow of water into the mine, a brief description is necessary of the topography and structural geology of the surrounding district, together with a review of the prevailing climatic conditions and a detailed study of the drainage of the Old Dominion area.

Topography and Structural Geology

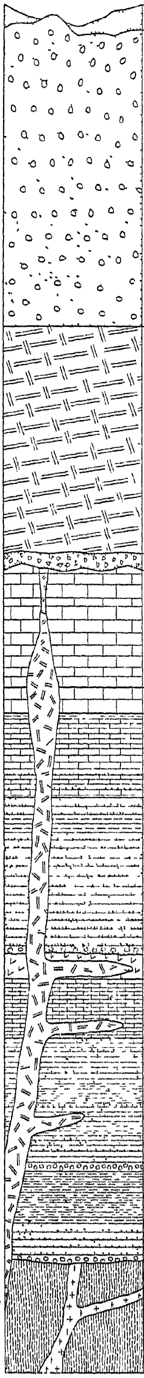
Globe is situated at an elevation of 3,500 ft. above sea level on Pinal Creek, the main drainage channel of the district, which rises in Pinal Mountains and flows on a very even grade northwest to where it joins the Salt River.

The topography surrounding Globe is irregular, but consists in general of a broad valley of rolling and slightly hilly lowland country, rising on either side of Pinal Creek and stretching to gradual rising foothill country, which in turn to the southwest flanks Pinal Range, the main mountain range of the district (with an elevation of 7,850 ft. at its highest peak), and to the northeast is terminated by the less steep and less conspicuous Apache Mountain range. Almost the entire portion of the valley zone to the west of Pinal Creek and country east of it south of the town of Globe, lies in Gila conglomerate, the most recent gravel formation, while north and east of Pinal Creek, in the immediate vicinity of the Old Dominion and United Globe mines, the surface formation consists of a much faulted and geologically complex area of limestones and quartzites intruded by diabase, mainly represented by a mammoth sill interlocated between the Cambrian "Mescal" limestone and the upper Cambrian or "Troy" quartzite and by several small dikes of quartz monzonite porphyry which also cut the diabase. These formations dip gently to the southwest toward Pinal Creek, and west of the main "A" shaft are overlain by a flow of dacite, which is in turn covered by the more recent Gila conglomerate to the west of Pinal Creek.

In the area northeast of the main "A" shaft, the less easily eroded quartzites largely form the capping of the ridges of the surrounding hills, while the gulches and basins are carved in the comparatively easily weathered diabase, the gulches often following the fault zones.

The lower foothills of the Pinal Range, south and southwest of the mine, change from the conglomerate of the valley or trough zone to the pre-Cambrian Pinal schist of the main range, large areas of which, intruded by granitic rocks, stretch to the northwest of the quadrangle. A columnar section of the geologic formations of the district is shown in Fig. 1.

The main "A" shaft of the Old Dominion mine is sunk in the foot wall



Unconsolidated Sands		Wash 0'-50'	Recent
Conglomerate Unconsolidated and Partially Consolidated Sands and Gravels		Unconformity Gila Conglomerate 1200' ±	Quaternary
Dacite Flow with Basal Tuff		Dacite 750' ± Unconformity	Tertiary
Coarse Angular Detritus		Whitetail Formation 0-75'	Tertiary
Limestones Light Gray, Dense Fine Grained Limestone Massive Bedding Fossiliferous.	Cut by Diabase	Great Unconformity Quarry Limestone 400'+	Upper Carboniferous
Limestones Hard, Crystalline, Gray to Black		Cliff Limestone 80'-100'	Lower Carboniferous
Limestones Thin Bedded, Buff Limestone with Shaly & Sandy Members Fossiliferous.		Pinto Limestone 300'	Devonian
Quartzites Medium to Coarse Grained, White Quartzite with some Fine Conglomerate and Argillite Coarse Conglomerate of Varying Thickness at Base		Upper Quartzite "Troy" 500'-700' Unconformity	Cambrian
Basalt Flow		Cambrian Basalt 50' ±	
Limestones Thin Bedded, Impure, Magnesian Limestone with Cherty Bands		Cambrian Limestone "Mescal" 200'	
Quartzites Thin Bedded Sandstones Quartzites and Argillites with more Massive Pink Quartzite at Base.		Dripping Springs Quartzite 425'	
Pebbles of Quartz & Quartzite		Barnes Conglomerate 20' ±	
Shales Red & Maroon Shales with Quartzitic Members Toward Bottom. Pebbles of Quartz & Feldspar		Pioneer Shale 275' Scanlon Conglomerate 1'-6'	
Crystalline Schists Cut by Granitic Rocks and Diabase		Great Unconformity Pinal Schist	Pre-Cambrian

FIG. 1.—COLUMNAR SECTION SHOWING GEOLOGIC FORMATIONS, GLOBE DISTRICT.

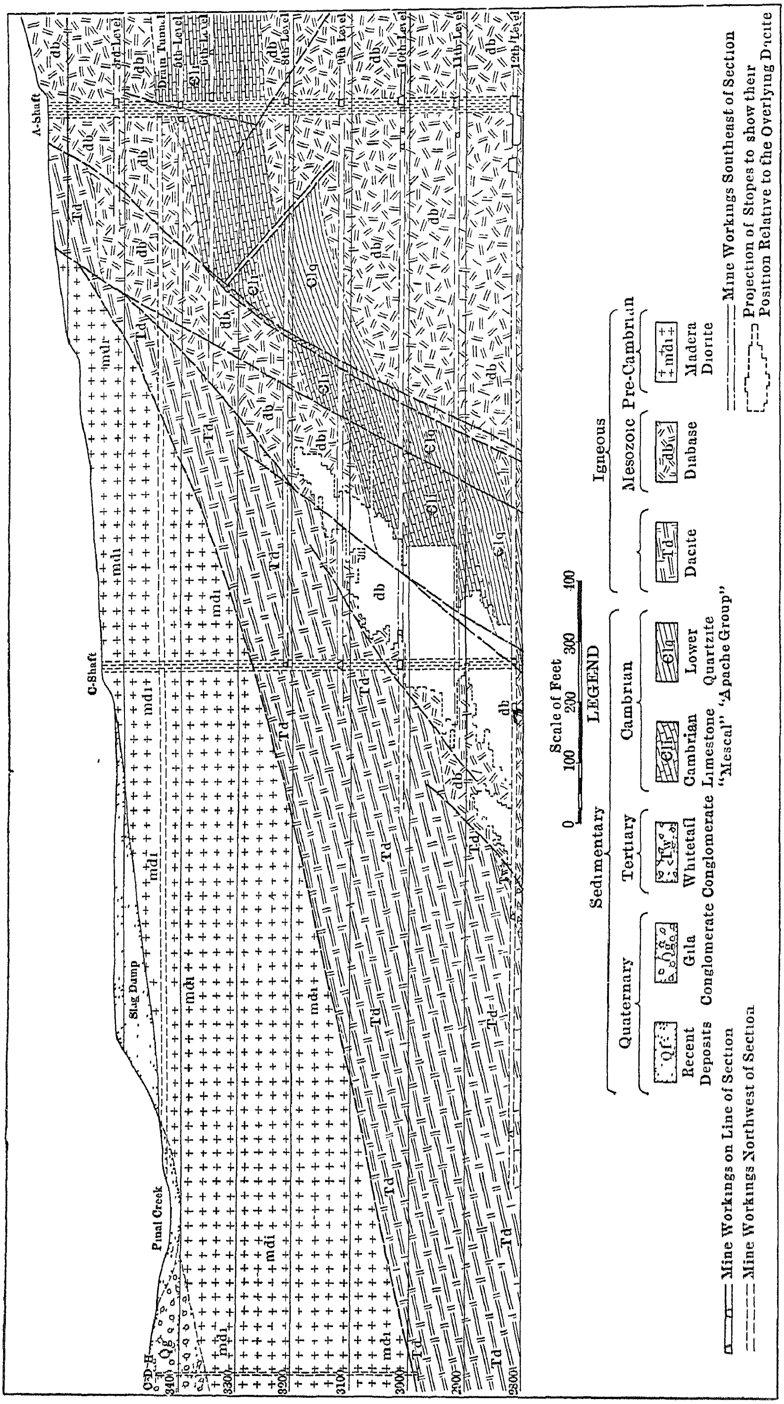


FIG. 2.—SECTION OF STRUCTURAL FORMATION FROM MAIN SHAFT WEST TO PINAL CREEK. (Section on A-A, N57° 47′E, Figs. 3 and 4.)

of the Old Dominion vein, a fault zone with a northeast-southwest trend, and from which, occurring in the form of big lenses or shoots, the principal ore extraction has come.

Southwest of the "A" shaft, as has been described before, the sedementaries dip gently to the west and are covered by a lava flow post-Mesozoic in age, which increases in thickness from its taper point, where it overlaps the older formations near the "A" shaft to some 350 ft. directly below Pinal Creek, thus effectively covering all trace of the ore zone in its westward trend. The Quaternary Gila conglomerate, as now exposed, shows some 2,000 ft. west of the "A" shaft. This formation also gains in thickness toward the west, and reaches a thickness of 1,000 ft. a short distance southwest of Pinal Creek. Between the dacite and upper gravel formation, there occurs a mass of Madera diorite, which was probably thrust-faulted into its present position in post-dacite time. Its thickness directly below Pinal Creek is somewhat over 300 ft. Fig. 2 shows a section of the structural formation from the main shaft west to Pinal Creek, and clearly indicates the relation of these superimposed non-mineralized formations to the underlying orebodies and to the formations favorable for ore exploration.

The Gila gravel formation in character ranges in size from fine sands through coarse gravel and pebbles up to big boulders. The pebbles comprising the conglomerate are more often angular than round, and where this formation has been cut, fragments are disclosed of all the older rocks of the area including schist-granite, diorite, limestone, quartzite and diabase formations. Its bedding is usually fairly well defined. The Madera diorite underlying it, in texture somewhat resembles a decomposed granite. Wherever encountered, it has been highly shattered, and where exposed at the surface is decomposed and crumbly in character, and from very recent investigation, intensely porous. The dacite flow immediately underlying the sedimentaries (omitting a thin bed of gravel formation accumulated over the sedimentaries before the outflowing of the lava) is a tough, comparatively firm rock when dry, and although it is the formation in which the water is always encountered in the western portion of the mine, is not in itself particularly porous, as proved by the not inconsiderable amount of work that has been done in it without encountering water. But once fractures in the dacite allow the downward seepage of water from the diorite above, the formation loses all pretense of firmness, absorbs water like a sponge, and becomes sandy, crumbly, and difficult to hold.

Climatic Conditions

Climatically and physically, the Globe District is typical of the mountain section of Arizona. In an ordinary normal year, the annual precipitation averages only 17 in. Most of this precipitation falls in the

summer months of July and August in the form of sudden and violent cloudbursts, which, while bringing immense quantities of water down the main drainage channels, is not of the character of precipitation that sinks very deep into the sun-parched ground. Winter rains, when they do come, are of the heavy, soaking type.

The surface of the ground throughout this area is mostly rocky and destitute of much soil. Hence, the district is largely barren of vegetal and tree growth, and, except on the higher flanks of the mountain ranges, where melting snows and deeper soil prolong tree life, and in special irrigated sections, the district physically is arid in the extreme.

Drainage Area of Old Dominion Mine

1. *General Description of Drainage Area*

The particular portion of Pinal Creek drainage area in which the mine is situated does not exceed 40 square miles, and may be considered as stretching from 3 miles south of Globe, where the watershed drains into the San Carlos Basin, a tributary of the Gila River, to the line where the northern limit of the company's property crosses Pinal Creek. The rainfall as registered at Globe for the years 1910 to 1914, and for the first six months of 1915 is shown in Table I. It is important to note, that of

Table I.—*Total Rainfall Registered at Globe, Ariz., for Years* 1910 *to* 1915

	Inches
1910	10 72
1911	21.36
1912	17.27
1913	14.72
1914	23.47

Last 6 Months, 1914

	Inches
July	4.90
August	3.17
September	0.47
October	3.24
November	1 40
December	5.59
	18.77

First 7 Months, 1915

	Inches
January	4 52
February	2.92
March	1.10
April	2.15
May	0 61
June	0.25
July	2 70
	14.25

the 23.4 in. of precipitation in 1914, 18.77 in. was recorded in the second six months, making a total of 30.3 in. from July 1, 1914, to June 30, 1915, or nearly double the normal precipitation for 12 months. No accurate figures are available on the run-off from the special area under consideration, or indeed, from the entire length of the Pinal Creek and Miami wash

drainage area, but it may be taken for granted that it varies considerably with the nature of the rainfall; *i.e.*, whether the latter originates in the form of cloudbursts in the mountains or as a steady downpour over the entire watershed. Also, it is largely dependent on the condition of the ground surface and the creek bed in the drainage area. It was very noticeable during the past winter, when the total monthly precipitation in December, January, and February was not only in the form of steady, soaking rains, but also as periodical heavy floods, that the water in Pinal Creek flowed steadily without cessation during the entire winter months, and increased in the spring by the late melting snows in the Pinal Range ran as a good-sized stream, carrying between 20,000,000 and 40,000,000 gal. per day, well into the month of June, an unmistakable criterion of the saturated condition of the surrounding ground.

Well measurements in the district showed that the water level in certain formations rose as much as 40 ft. in three months, the rise being very rapid during the latter period of the heavy rainfall, and apparently taking place after the ground had absorbed all the previous moisture possible. Recent weir measurements, taken at the Old Dominion weir at the south end of its property, showed at the height of a summer flood, when over 2 in. fell in 8 hr., that as much as 300,000 gal. per minute was flowing in the creek.

2. *Different Sources of Water in the Mine*

Owing to the peculiar geological conditions previously described, the Old Dominion has, in its problem of handling water in the mine, two distinct sources to contend with; one the east-side water, and the other the west-side water.

The east-side water is that coming into the lowest levels of the mine, and is doubtless the result of the lowering, in the vicinity of the mine workings, of the normal ground-water level of the entire district. The east-side water is encountered principally in shale and quartzite formations, and rarely in the denser diabase. The driving of each successive lower level drains the water from the level above. It has a normal temperature of 90° F., or 25° warmer than the west-side water. While this flow is of considerable importance, it is fairly constant in quantity, averaging about 2,000,000 gal. per day and is not subject to the variations of the west-side water. The east-side water source, therefore, will not be discussed in detail in this paper.

The west-side water is the water held above the dacite formation in the west end of the mine, and while this water is also ground water, it is prevented from sinking lower in common with the general drainage of the mine area by the impervious layer of dacite that separates it from the underlying mine workings. As a consequence, although mining opera-

tions are proceeding on the 18th level in the west section of the mine, the lowest drainage point for this west-side water is on the 12th level, 600 ft. above, where the water is tapped through fractures in the dacite. This water contains a relatively small proportion of solids, and is excellent for domestic purposes. Inasmuch as the handling of the west water has been in recent years the most troublesome of the mine's pumping problem, this particular source will be considered in detail.

3. *Description of Underground Workings Adjacent to the West Drainage Source*

As a matter of recent history, and as leading up to an account of the disastrous inrush of water on the 12th level in March, 1915, it may be stated briefly that as the mining of the orebodies, to their western limits immediately under the dacite capping, proceeded on the upper levels, so did the necessity of draining the water contained in and above the dacite to a continually lower level become more evident. Twice before the latest inrush of water was the safety of the mine imperilled by the sudden and unexpected tapping of large volumes of water held above the dacite, which were encountered while mining ore along the dacite contact.

In 1906, the breaking in of the water above the 10th level drained within a very short time the water on the 8th and 9th levels above, and it was deemed necessary to install adequate pumping facilities on the 12th level and to drain all water to that level in order to extract the ore remaining above that horizon that lay close to the dacite formation. This was accordingly done, and the intake of the water lowered 200 ft. to the 12th level, which in turn, in a general way, but as will be shown later, somewhat imperfectly drained all of the country above. A plan of the 10th level is shown in Fig. 3. In 1914, therefore, the 12th level was the main drainage and pumping level for all the water originating on the west side of the mine, and also the main relay level for all water pumped from the center and eastern portions of the mine below that level. Fig. 4 shows a plan of the pertinent features on the 12th level from the shaft and pump station out to the original draining point. The water drift first run to carry the water back to the pump station is marked as *AA*, and shows the formation through which it passed before tapping the water well within the dacite. Later, on account of the original drift being directly above an orebody on the 13th level and the seepage of the water from the drift interfering with the stoping operations below, a new drift *BB* was run well into the foot wall and away from any possible ore below. At the same time, the old drift was dammed up.

During 1913 and 1914, active and continuous efforts had been made to drain what is known as the Zero East, and Zero West section of the mine marked in circles in Fig. 4. Considerable ore remained between the 12th

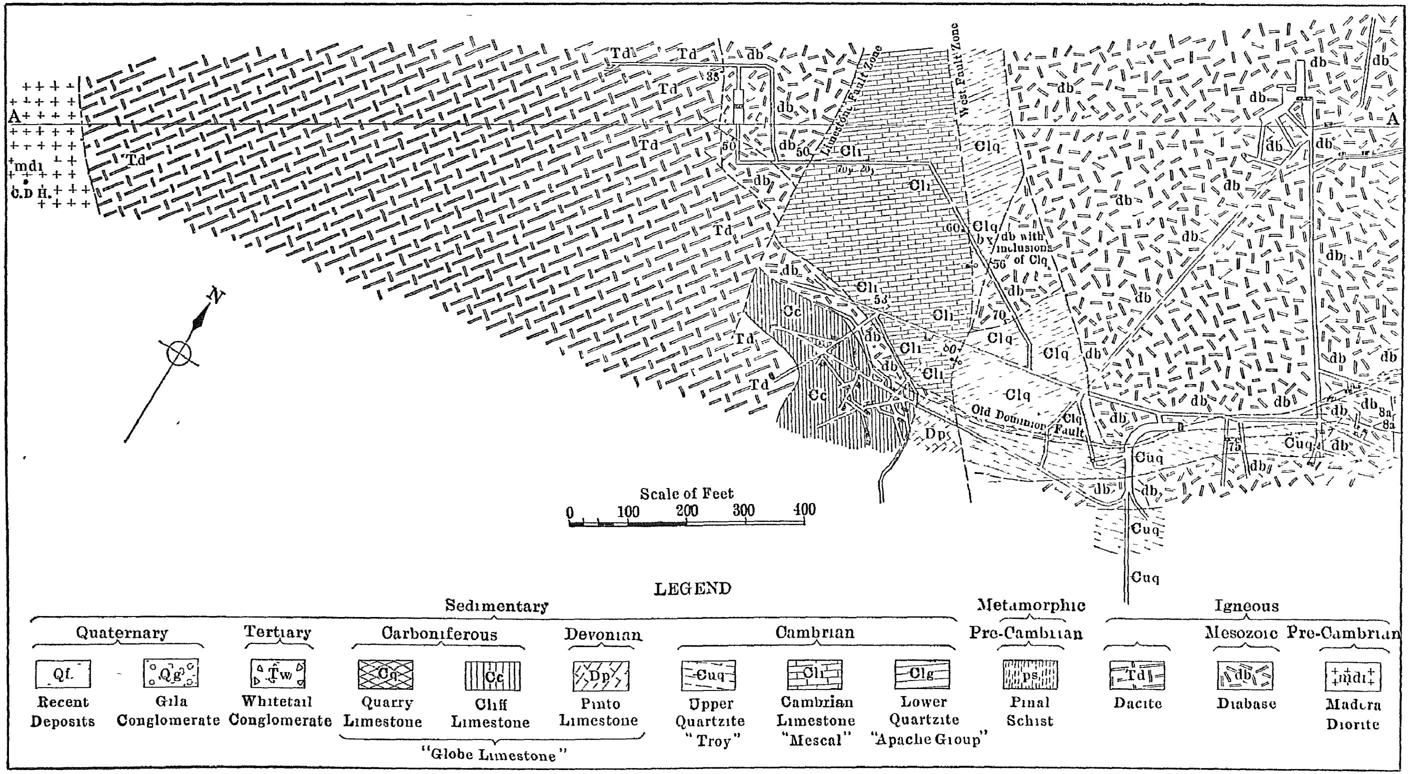


FIG. 3.—GEOLOGIC MAP OF PORTION OF 10TH LEVEL, OLD DOMINION MINE.

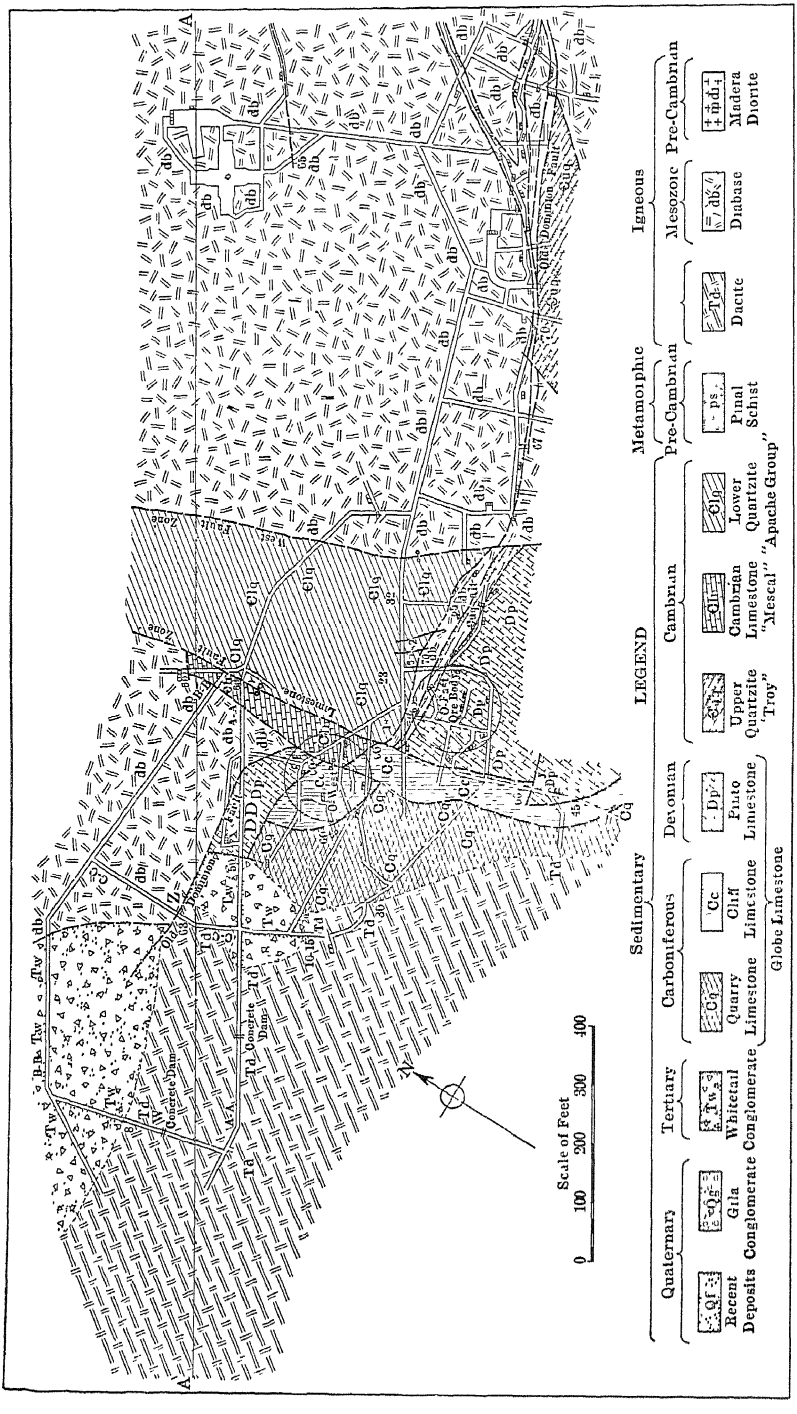


FIG. 4.—GEOLOGIC MAP OF PORTION OF 12TH LEVEL, OLD DOMINION MINE.

and 11th levels in this territory, but although a considerable amount of work had been done in the dacite itself in the immediate vicinity of the orebody, water was never encountered in sufficient quantities to drain the country above. But as an outlet for the water that was being produced by this section of country, a drift *CC* cutting across the old water drift was driven to the new water drift *BB*. This drift *CC* is of interest because at approximately the point *Z* the first inrush of water broke through last March. In short, previous to the inrush of new water, the drift at *BB* was delivering all of the water coming from the draining point in the dacite marked *W* in Fig. 4. A comparatively insignificant amount was coming through the drift *CC* from the undrained Zero country. The old water drift *AA* was dammed near its western end, but was opened east of the drift *CC* to opposite 13–1–11 raise marked *X* on the map. This raise was used for the purpose of carrying air from the *C* shaft to the 13th level. Below the 12th level ore was being extracted on the 13th, 14th, 15th, and 16th levels, each 100 ft. apart. Below the 16th level, the main shaft had been sunk to the 18th level, and pumps had been recently installed to handle the water on that level coming from the central and eastern portions of the mine.

General

In 1909, with the idea of eliminating any possible seepage of water from the creek bed into the mine, a concrete dam was put down to bedrock in Pinal Creek by the Old Dominion Copper Co. a quarter of a mile south of the mine. All water running in the creek was thus forced over the dam and by means of a flume was kept out of the creek for a distance of 2,300 ft.

Because no difference in the water flow in the mine was apparent when the water was bypassed, this practice was in time discontinued. Recent comprehensive but at present incomplete experiments with weirs and pipe readings of the loss of water through the sands and gravels taken at various points in the creek, also by measurements of the level of the standing water in the gravels over the various underlying formations, tend to indicate that perhaps the flume was not carried far enough down the creek to insure the carriage of the water past and over the most absorbent formations. Fig. 5 shows the annual water flow on or above the 12th level of the mine in the year 1913, together with the rainfall curve for those 12 months, and is typical of the recent normal year's water-pumping operations from the west side of the mine. In addition to the flow records on the chart as coming from the 10th and 12th levels, there was pumped from the lower levels of the mine up to the 12th level a fairly steady flow of between 1,500,000 and 2,000,000 gal. per day.

The flow diagram recorded in Fig. 5 seems to indicate that in normal years a small increase of water is recorded in the mine, following with a

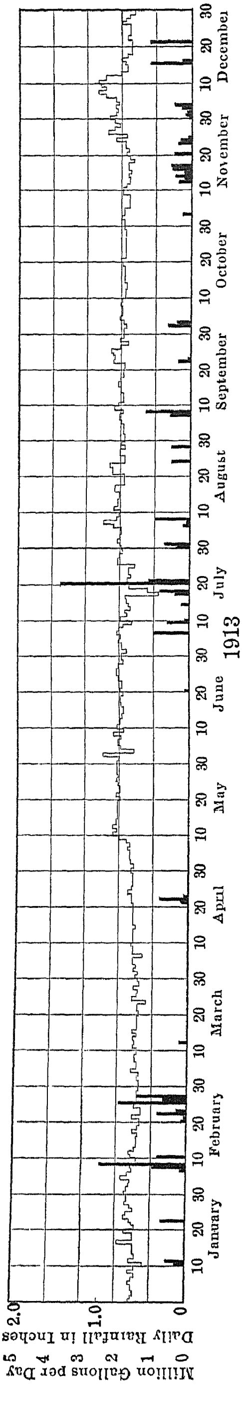


FIG. 5.—WATER FLOW ON AND ABOVE 12TH LEVEL AND RAINFALL IN 1913

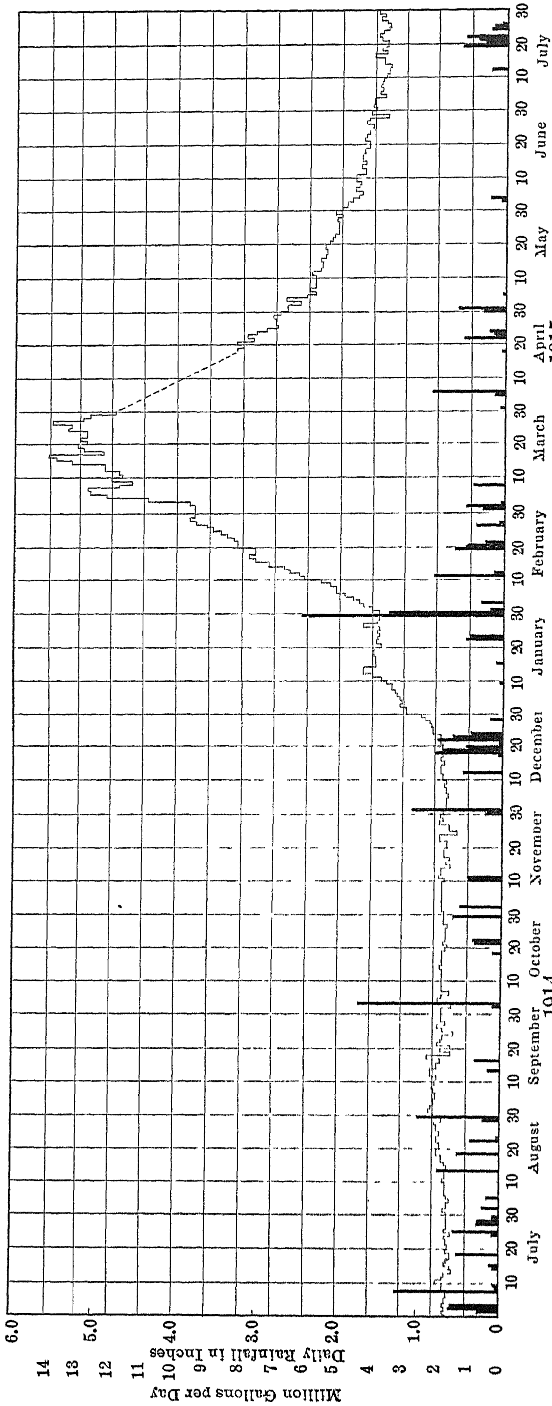


FIG. 6.—WATER FLOW ON AND ABOVE 12TH LEVEL, JULY 1, 1914, TO JULY 31, 1915, AND RAINFALL DURING SAME PERIOD.

lag of 6 or 7 weeks behind the summer rains. Fig. 6 shows the quantity of water pumped from the mine for the 13 months from July 1, 1914 to July 31, 1915, together with a curve of the rainfall during that period. It indicates how the period from July 1 to Dec. 15 was what may be termed the "saturation" period, and that although the precipitation was considerable during these months, there was no increase in the flow of water from the western portions of the mine. It indicates how the 5½ in. of rain which fell in December, after the ground had absorbed all the moisture possible, was quickly reflected by the sudden rise in the mine flow, but how apparently following on a period from Jan. 1 to Jan. 20 of comparatively little precipitation, the mine water showed a tendency to decrease. The precipitation of Jan. 22 to 24 was, however, immediately felt in the mine, and the additional 5 in. of rain that fell on a few consecutive days at the end of January and beginning of February caused the mine flow to increase steadily until the middle of February. A few days drop at this time when the 10th and 12th level flow amounted to over 7,750,000 gal. and the total amount pumped to over 10,000,000 gal. per day, led us to hope that the worst had been reached. But following a rainfall of 1½ in. about Feb. 20, the water flow increased steadily on the 10th and 12th levels until Mar. 4, when the "break" came in the mine and we handled in excess of 14,000,000 gal. on that day.

III. PUMPING EQUIPMENT

Normal Equipment

1. *Preliminary*

The normal pumping equipment of the Old Dominion consists of steam pumps on the 10th, 12th, and 14th levels which pump direct to the water storage tanks on surface and to the drain tunnel which runs from a point in the main "A" shaft 235 ft. below the surface discharge level to Pinal Creek.

Electric pumps are installed on the 14th, 16th, and 18th levels, which are all approximately 200 ft. apart, and these pumps lift the water to the main pumping station on the 12th level, whence it is relayed to the surface or to the drain tunnel.

A boiler plant is situated adjacent to the main "A" shaft to generate steam for the steam-operated pumps underground, while electric power for the lower level pumps is transmitted from the central power plant near the smelter. The normal boiler equipment at the "A" shaft consists of five 335-hp. Stirling boilers, equipped with superheaters and economizers, and one 250-hp. Stirling boiler, while to these were added during the flood period one 250-hp. Scotch marine boiler and one 200-hp. boiler of the same type.

During the heavy water period in February, March, and April, compressed air was supplied for the air lifts and sinking pumps from the central power plant.

The "A" shaft, which is the main hoisting shaft for the property, consists of five compartments: Two skip hoisting, two cage hoisting, and one reserved for steam lines, air lines, water columns, and electric cables.

A pump winze runs from the 10th level down to the 18th level, and on the 11th level at the "A" shaft the water columns and main steam lines are transferred from the shaft to the pump winze, and thence down to the main pump station on the 12th level. All water columns from below the 12th discharging to that level are also brought up in this pump winze.

2. *Steam Lines*

The steam lines from the main boiler plant consist of one main line, 6 in. in diameter down to the 11th level and 5 in. thence to the 12th level; also one auxiliary steam line, 8 in. in diameter to the 10th level, 6 in. from the 10th to the 12th and 4 in. from the 12th to the 14th. A 5-in. line branches from the auxiliary line at the 12th level, and the two 5-in. lines are cross-connected at this point with the main lines down the shaft to allow the 12th level pumps being run off either steam line in case of repairs to the other. At the 10th level a 4-in. steam line is run to each of the triple-expansion pumps from the 8-in. auxiliary line, and at a steam receiver on the 12th level into which the two 5-in. lines connect, 4-in. headers are run to the four triple-expansion flywheel pumps.

The triple-expansion Prescott pump on the 12th takes its steam through a 4-in. line run from the auxiliary 6-in. line.

3. *Water Columns*

There are three 12-in. and one 14-in. water columns running from the 12th level up the surface, allowing discharge either at surface or at the drain tunnel. The 12-in. columns are of steel pipe with cast-iron flanges; the 14-in. column is wood lined. A fifth water column, 10 in. in diameter, was put in during the high-water period from the 10th level up to the drain tunnel. Below the 12th level two 10-in. water columns run from the 18th level up the pump winze, which columns are cross-connected at the 16th level with the electric pumps on that level. A third 10-in. water column discharges from the electric pump on the 14th level to the 12th pump station. Fig. 7 shows a diagram (not to scale) of the water columns and steam lines and their connections to the various pumps.

The 12th level is 197 ft. below the 10th level; the 10th level is 436

ft. below the drain tunnel, which in turn is 235 ft. below the point of discharge into the surface water-storage tanks. In other words, the 12th level, pumps in discharging to surface, have a lift of 868 ft. and when discharging at the drain tunnel, a lift of 633 ft.

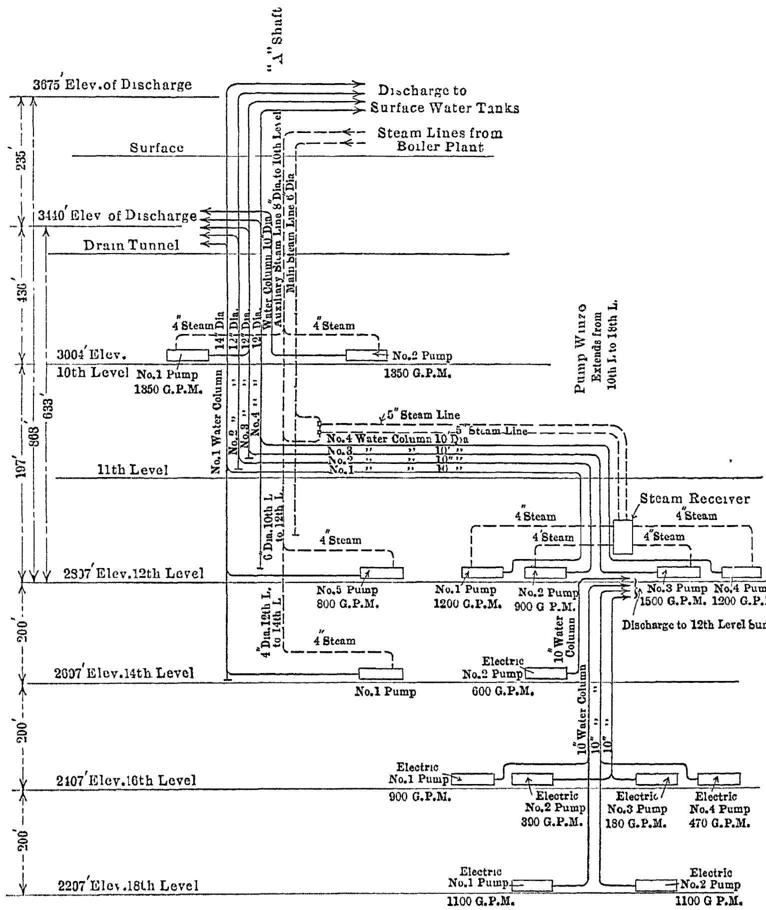


Fig. 7.—Diagram of Water Columns and Steam Lines with Connections to Pumps.

4. *Pumping Machinery and Pump Stations*

On the 10th level are two triple-expansion duplex type, 15 by 27 by 39 Prescott pumps, capable at the lift of 436 ft. to the drain tunnel of delivering 1,350 gal. per minute. On the 14th level there is a similar type pump which at the lift of 833 ft. to the drain tunnel is capable of delivering 800 gal. per minute.

On the 12th level there are four triple-expansion flywheel pumps

14–26–26–26 with 36-in. stroke, of Nordberg manufacture, capable at 56 r.p.m. of delivering from 900 to 1,500 gal. per minute, according to the size of the water plungers, which vary in the different pumps from 6$\frac{5}{16}$ in. in pump No. 2 up to 8$\frac{1}{4}$ in. in pump No. 3.

The triple-expansion Prescott pump on the 12th level is 15 by 27 by 39 with a 24-in. stroke. This pump is used as a reserve and at a speed of 35 r.p.m. can deliver 800 gal. to the drain tunnel.

These steam pumps are run condensing. Steam pressure at the pumps averages about 134 lb. and the steam temperature at the pumps is approximately 350°.

Recent tests run under operating conditions show the steam consumption by the triple-expansion flywheel pumps, which are the ones used in normal times, to be 18.0 to 18.2 lb. per water horsepower-hour.

Table II.—*Old Dominion Copper Mining & Smelting Co.*

Mine Pumps

Pump Number	Location, Level	Type of Pump	Steam Cylinders, Inches	Motor Horsepower	Water Plunger, Inches	Stroke, Inches	Speed, Revolutions per Min.	Capacity per Min., Gal	Lift, Feet	Location of Discharge
1	10th	Triple-expansion duplex	15–27–39	....	12½	24	30	1,350	436	Drain tunnel
2	10th	Triple-expansion duplex	15–27–39	...	12½	24	30	1,350	436	Drain tunnel
1	12th	Triple-expansion flywheel	14–26–26–26	....	7$\frac{5}{16}$	36	56	1,200	Drain tunnel 633 Surface 868	Drain tunnel and surface.
2	12th	Triple-expansion flywheel	14–26–26–26	....	6$\frac{5}{16}$	36	56	900		
3	12th	Triple-expansion flywheel	14–26–26–26	....	8¼	36	53	1,500		
4	12th	Triple-expansion flywheel	14–26–26–26	....	7$\frac{3}{16}$	36	56	1,200		
5	12th	Triple-expansion duplex	15–27–39	....	9¼	24	35	800	633	Drain tunnel
1	14th	Triple-expansion duplex	15–27–39	. .	9¼	24	35	800	833	Drain tunnel
2	14th	Vertical triplex, double-reduction	. . . . .	70	12	16	30	600	200	12th level
1	16th	Vertical quintuplex, single-reduction.	. .	150	8	18	51	900	400	12th level
2	16th	Vertical triplex, double-reduction		42.5	10	16	28	390	400	12th level
3	16th	Vertical triplex, double-reduction	. . . .	40	7	10	42	180	400	12th level
4	16th	Vertical triplex, double-reduction	. . . . .	70	10	16	34	470	400	12th level
1	18th	Horizontal quintuplex, single-reduction.	....	250	9	18	49	1,100	600	12th level
2	18th	Horizontal quintuplex, single-reduction.	. . ....	250	9	18	49	1,100	600	12th level

In normal periods, any water on the 10th level is siphoned to the 12th level through the "C" shaft, which makes the latter level the highest pumping point for all water. Of the four flywheel triple-expansion pumps on the 12th level, No. 2 takes its water through a pipe from the concrete dam at the end of the 12th level water drift, and this water is

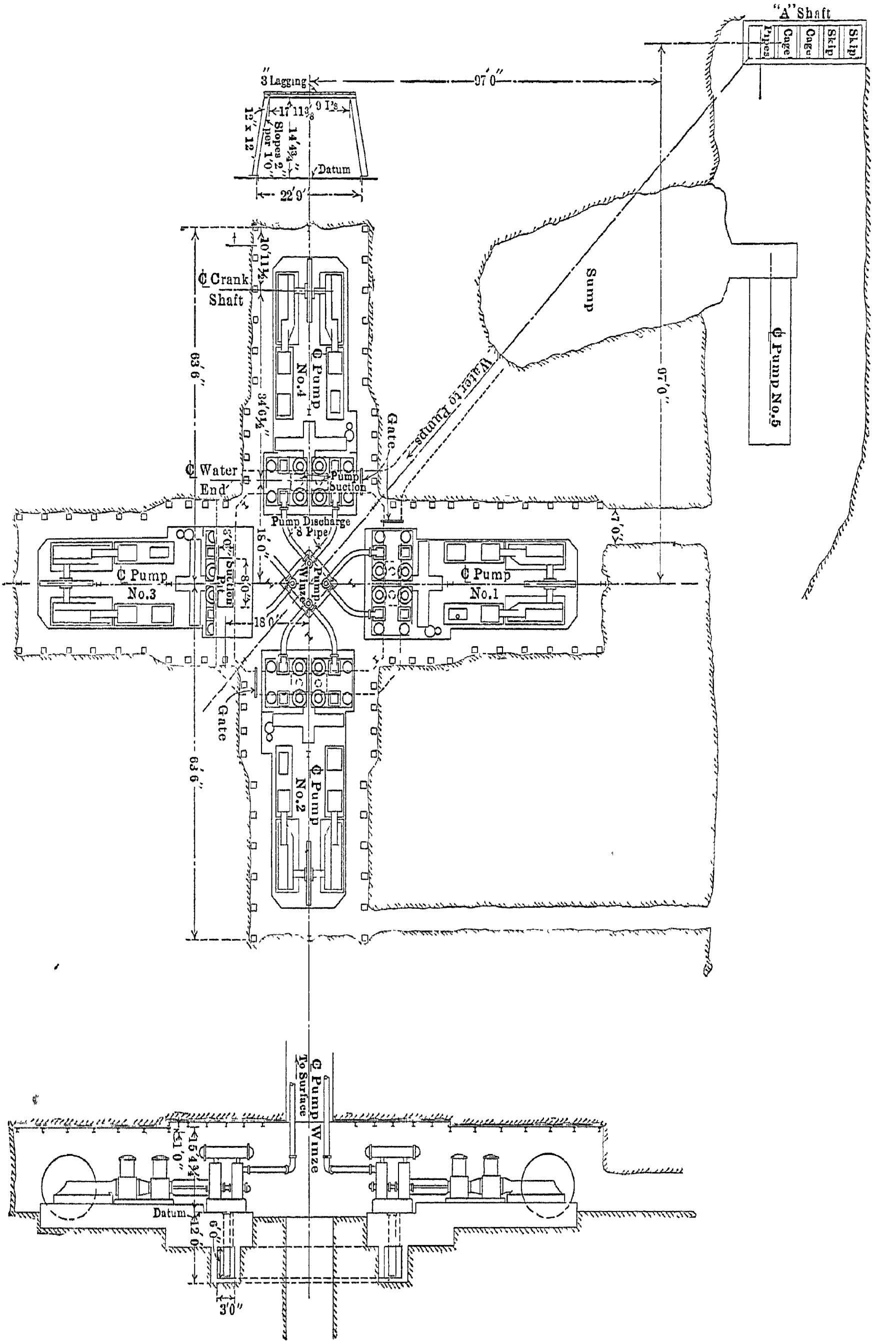


Fig. 8.—Arrangement of Pump Station on 12th Level.

used for domestic and boiler purposes. Nos. 1, 3, and 4, together with the triple-expansion Prescott (when in use), take their water direct from the station sumps, and pump either to surface or to the drain tunnel, according to where the water is needed. The water necessary for the concentrator and fire-line purposes goes direct to surface and the surplus to the drain tunnel. A fixed quantity of this surplus is delivered daily to the Miami Copper Co. while the remainder runs to waste in the creek. Below the 14th level the pumps are all electric driven; on the 16th there is one vertical quintuplex, single-reduction, Aldrich pump with 150 hp. alternating-current motor. This pump lifts at the rate of 900 gal. per minute from the 16th to the 12th level. Housed in the same station with the quintuplex pump are three vertical, triplex, double-reduction pumps, varying in size and capacity, and run by direct current. The capacity of the biggest of these triplex pumps is 470 gal. per minute, and of the smallest, 180 gal. per minute. These three pumps also lift from the 16th to the 12th level. On the 18th level, 200 ft. below the 16th, there are two horizontal, quintuplex, Aldrich pumps, both with 250-hp. alternating-current motors. The capacity of both these pumps is 1,100 gal. per minute from the 18th to the 12th level. In Table II is given a list of the pump equipment which was being used at the end of 1914, and to which, as will be shown later, were added many small emergency pumps and air lifts during the flood period.

The arrangement of the pump station on the 12th level is somewhat unusual and of interest. It is in the form of a maltese cross, as reference to Fig. 8 will show. In the center of the cross is the pump winze which, as has been noted previously, is used for pump columns and steam lines from the 11th to the 18th level. Each arm makes, in a manner, a separate station, and is 63 ft. 6 in. long, measured from the center of pump winze. The width of each arm of the station is 22 ft. 9 in. at the floor line; and the height is 14 ft. 4¾ in. Each of the four arms of this cruciform station houses one of the triple-expansion flywheel pumps with its discharge end set toward the pump winze in the center, and each having its own discharge column. This particular arrangement has considerably simplified the piping in the station, by means of short and easy sweeps into the main water columns. The way in which this station has been laid out is very satisfactory, but doubtless costs more for excavation than a plain rectangular one of the same cubic footage. A sump which holds in the neighborhood of 80,000 gal. is situated between the station and the shaft. The sump discharges the water into a concrete conduit supplying the suction pits of the four pumps. This conduit has doors in it, permitting the shutting down of any one suction pit for cleaning or inspection. The conduit is 12 ft. below the pump station floor line and is 3 ft. wide. The intake is screened to prevent wood chips and waste getting to the valves.

Emergency Equipment

1. *Auxiliary Pumps*

Although several small emergency pumps were installed on the 12th and 10th levels toward the end of February when the regular pumping equipment was reaching its limit, these emergency pumps were only used for a comparatively short time, and were later replaced by air lifts on account of the latter's much greater pumping capacity and freedom from shutdowns.

Two 14 by 12 by 8 Prescott sinking pumps were installed on the 10th level toward the end of February. They helped out the two triple-expansion pumps on that level when the latter were reaching their limit, while on the 12th level there was installed a 500-gal. duplex piston pump and one 14 by 12 by 8 Prescott sinker. These emergency pumps were all operated by air.

2. *Air Lifts*

Early in March an air lift was installed from the 12th to the 10th level and later one from the 10th to the drain tunnel. The submergence on the air lift from the 12th to the 10th level was 177 ft., or 47 per cent. The top 100 ft. of the column was made of 12-in. pipe, the bottom 100 ft. and the two legs were of 10-in. pipe. The air was supplied from a 4-in. line at 80 to 90 gage pressure.

Table III.—*Tests of Air Lifts. Old Dominion Copper Mining & Smelting Co.*

Test No.	Net Lift in Feet, Excluding Friction	Submergence in Feet	Submergence, Per Cent.	Gage Pressure of Air in Lb.	Amount of Opening in 4-in. Air Valve	Water Pumped by Air Lift. Gal. per Min.	Free Air Used by Air Lift. Cu. Ft. per Min.	Cu Ft. of Free Air Used per 1,000 Gal. of Water	Efficiency = Water Hp at Air Lift to 1 Hp. at Compressor, Per Cent.	Pounds Steam Used at Compressor per Water Horsepower-Hour
A–1	200	177	47.0	80	¼ turn	1,011	1,353	1,338	28.8	70.8
A–2	200	177	47.0	80	½ turn	1,679	1,809	1,080	36.0	50.3
A–3	200	177	47 0	80	1 turn	1,793	2,262	1,261	30.9	58 8
A–4	200	177	47 0	80	1¼ turn	1,925	2,658	1,375	28.4	64.4
A–5	200	177	47 0	80	Full	1,965	3,219	1,638	25.5	71.6
B–1	431	188	30.4	90	1 turn	1,317	3,484	2,645	32.9	57.7
B–2	431	188	30 4	90	2 turns	1,291	3,832	2,968	27 0	64.8
B–3	431	188	30.4	90	3 turns	1,291	3,919	3,035	26.4	66.2
B–4	431	188	30.4	90	Full	1,325	4,089	3,086	26 0	67 2
C–1	431	188	30.4	90	1 turn	1,122	3,051	2,718	29.4	59 5
C–2	431	188	30.4	90	2 turns	1,233	3,306	2,681	29.9	58.4
C–3	431	188	30.4	90	Full	1,233	3,484	2,825	28.4	61.3

Tests in group "A" were on the air lift pumping from 12th level to 10th level. Tests in group "B' and group "C" were on the air lift pumping from 10th level to drain tunnel.

Short tests on this air lift showed that it handled to the 10th station over 1,650 gal. per minute, with an air consumption of 1,080 cu. ft. per thousand gallons. The highest efficiency obtained was 36 per cent.

In the lift from the 10th up to the drain tunnel, which was 431 ft.,

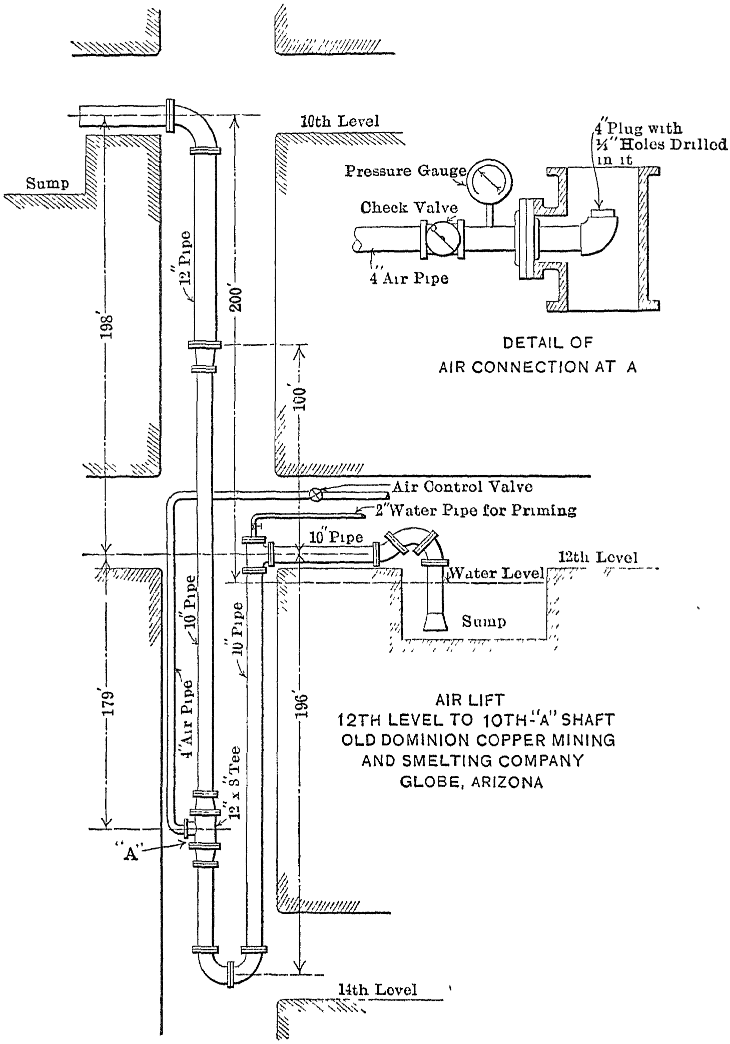


FIG. 9.—ARRANGEMENT OF PIPING ON 200-FT. AIR LIFT.

the submergence was 188 ft., or 30.4 per cent. The top 200 ft. of the column was 12-in. pipe, while the bottom 231 ft., and the two legs were of 10-in. pipe. The air was supplied through a 4-in. pipe at a gage pressure averaging from 90 to 100 lb., but which was always kept as high as possible. This lift delivered to the tunnel 1,233 gal. of water per

minute, with an air consumption of 2,681 cu. ft. per thousand gallons of water delivered. The maximum efficiency showed practically 30 per cent.

During the period of unwatering the 16th level, the two columns on the 18th level Aldrich pumps were converted into air lifts; one delivering to the 14th level and the other one to the 12th. The water at that time stood 36 ft. above the 16th level. These lifts were made up as follows: A 3-in. air line provided with a foot-piece made from 3-in. steel tubing, plugged at the bottom and perforated with 3/16-in. holes drilled at a 45° angle, was lowered to the sweeps connecting the columns with the pumps. Both lifts pulled the water through the valves and suction of the Aldrich pumps, and delivered to the 14th level about 1,300 gal. of water per minute, and to the 12th about 1,000 gal. per minute. Fig. 9 gives a sketch of the arrangement of the piping used on the 200-ft. air lift, and Table III shows the result of short tests run on these air lifts.

IV. HIGH-WATER PERIOD

Increased Flow into the Mine

The month of December, 1914, in Globe, was the wettest month of the year, over 5½ in. falling in the 31 days, 4 in. of which fell in the 8 days from the 17th to the 25th. The flow of water on the 10th and 12th levels West was about normal to the 20th of the month, and apparently the previous rainfalls had not made themselves felt, but on the 20th day the water originating on the 10th and 12th levels, and which was at that time all handled on the 12th, began to increase from the normal flow of 1,750,000 gal. for 24 hr. until on Jan. 1, we were handling over 3,500,000 gal. per day coming from the 10th and 12th levels West. This, with the more or less constant amount of 2,000,000 gal. from the 16th and 18th levels which was being relayed from the 12th level, totalled a flow of 5,500,000 gal. per 24 hr. to be handled from this latter level. By the middle of January, 6,500,000 gal. per day were being pumped from the upper and lower levels.

The East water on the 18th level showed no increase whatever during the period the water was rising on the 10th and 12th levels.

Toward the end of January, in order to relieve the 12th level pumps, one of the triple-expansion pumps on the 10th was started, and two weeks later its duplicate on that level was put into commission.

By the middle of February the water was showing in the "C" shaft above the 9th level and the two 10th level pumps were pumping 3,200,000 gal., while the four flywheel triple-expansion pumps on the 12th, aided by the triple-expansion Prescott on that level were pumping 7,000,000 gal. more. All of these seven big pumps were operated by steam gen-

erated at the "A" shaft boiler plant, and the entire battery of six boilers was running overload.

Feb. 26 saw the total water pumped from the mine pass the 12,000,000 mark. The central power plant provided air for additional emergency pumps on the 10th and 12th levels, and a hurriedly installed 250-hp. boiler at the "A" shaft helped to relieve the overtaxed normal steam generating equipment at that point. By the last day of February, the mine had pumped 270,000,000 gal. in the 28 days of the month, and there was no sign of the water flow slackening. On Mar. 3, the day before the inrush of water, the total amount rose to 12,600,000 gal.

Such, in brief, is the story of the rise of the water to its high point.

Inrush of Water on 12th Level, Causing Flooding of 18th and 16th Levels

1. *Account of "Break" on 12th Level*

It became obvious in the first few days of March, that with the water increasing steadily on the 10th and 12th levels, and coming down the "C" shaft in ever greater volumes from the 9th level and above, that we had about reached the limit of our available pumping and power capacity, and our only hope was that the flow had reached its height and would gradually decrease. But what actually happened exceeded our most pessimistic anticipations.

On the afternoon of Mar. 4, word was telephoned up to surface that a big flow of water had broken into the new water drift on the 12th level and that the pumps on that level were unable to handle it. A hurried inspection of conditions on the 12th level showed an immense torrent of water coming surging out to the pump station. With all five big pumps running overspeed, the pumping equipment was absolutely unable to cope with the flow. The sumps were overflowing and the water rushing down the pump winze and main shaft to the 18th level. A realization of the volume of water made it clear that the 18th level was doomed to be drowned very quickly. As the 12th level pumps are relays for the 18th level water, it was obvious that if the former were already overtaxed, that it was useless sending any water up from below. Within 30 min. the 18th level quintuplex electric pumps were under water, and by 7 o'clock that evening, the water had risen in the pump winze and main shaft to the 16th level. The motors of the four electric pumps on the 16th level were left in place just as long as possible in the hope that the flow on the 12th would subside sufficiently to save these from loss. When, however, the water started creeping up on the station, hurried attempts were made to get the motors off, but the rise of the water, caused by the overflow from the 12th level, was so rapid that within a very short while the work

had to be discontinued. All equipment possible that could be moved, in the form of locomotives, tools, and cars, was transferred to the east side of the mine, 3,000 ft. away, where the water had not yet reached. The torrent of water coming down the main shaft was of such intensity that it became unsafe to operate the cages below the 12th level in the "A" shaft, which is the only operating shaft sunk below the 12th level. In the meantime, a considerable flow of water commenced showing on the 14th level West. This necessitated starting the triple-expansion Prescott pump on that level, which throws direct to the drain tunnel. By 8 p.m. every available pump in the mine above the 16th level was running over-speed, and the water was overflowing from the 12th and 14th levels down to the 16th level, which latter level by late that night was completely submerged to its farthest point east and west.

2. *Serious Problems Caused by Inrush of Water*

A thorough inspection of conditions in the water country on the 12th level showed that the two most serious matters to be coped with at once were, first, the concentration of all the water possible on the 12th level, and second, the handling of the immense quantity of sand that came surging out with the water. The inspection revealed that the new rush of water that was causing all the damage was coming in from the crosscut marked *CC* on Fig. 4 that connected the old and new water drifts. But how far back toward the old water drift the "break" had come was impossible to tell, owing to the immense quantity of débris, varying in size from big boulders down to fine dacite sand, that was being deposited in the crosscut and in the new water drift. As the crosscut became blocked with débris, and its outlet, to a partial extent, dammed, the water backed up and found its way along the old water drift to 13–1–11 raise which was used as a ventilating raise for air coming from the "C" shaft. The water was running down this raise to the 13th level and thence through several of the stopes to the 14th level. Reference to Fig. 4 will make the location of these various points clear. It was recognized at once that it was vitally necessary to prevent the water flowing down below the 12th, where, not only were the pumps insufficient to handle any continued large volume of water, but where the water could do great damage to the orebodies. It was, therefore, essential that the raise should, if at all possible, be closed or dammed up without delay. The only possible way into the top of the raise would have been through the drift *DD* from the point marked *Y*. Here it was found that the débris and sand had come out in such quantities that the drift was completely filled with dacite sand, brought down by the water to within a few feet of point *Y*. This being fully 300 ft. from the raise made it out of all question to reach the objective point from that direction. A similar

filling up of all drifts adjacent to the bottom of the raise on the 13th made it equally impossible to reach the raise within anything like reasonable time on that level, since the removal of sand and rock by mucking out the drifts quickly proved futile, as it simply made room for the further deposition of sand. Being thus effectively cut off from reaching the raise, either at the top or bottom, our main hope was that it would soon choke and block up with sand, of its own accord. All efforts were, therefore, centralized in keeping open the main water drifts on the 12th level from the "A" shaft to the crosscut connecting the old and new water drift so as to insure at all costs a free exit for the water on the 12th, and to prevent any backing up behind the points where the water broke in with the consequent tendency to flow down the ventilation raise to the 13th and 14th levels.

This led up to the second big problem. The water was coming in periodic but tremendously strong surges, carrying immense volumes of sand and rock, the finer sand, despite hastily constructed dams, being carried into the sumps at the pump station. The rushes produced volumes of water and sand that, in addition to the flow already overtaxing the pumping capacity, caused the sumps to overflow continuously. These periodic heavy surges of water were doubtless occasioned by the continual damming up with sand and rock of the main channel through which the water came down from above the 12th level; when the pressure became too strong the water broke through with a rush and then ran steadily for a short time, only to get dammed up again with a repetition of the same performance. With a conception of a very recently formed channel through soft and caving formation, this procedure can easily be imagined. These constant surges and the débris they brought down were the most trying features in the control of the water.

Handling of Water during Flood Period and Unwatering of 16th Level

The night of Mar. 4 found us in the following unenviable position: Water was coming into the west end of the mine from four different sources—on the 10th level, in the vicinity of the "C" shaft, on the level, and from above the 10th level in the shaft itself at the rate of 3,000 gal. per minute, all of which was clear water without sand, and was handled by the 10th Prescott pumps and two sinking pumps.

On the 12th level clear water without sand was coming in from the original source at the concrete dam, while from the most recent source in the crosscut connecting the old and new water drifts the water was pouring out in a torrent into the drift, bringing with it immense quantities of sand and rock. This flow, combined with the clear water, was largely going out to the "A" shaft main pumping station. But there was also a portion of it flowing down the "C" shaft from the 12th to the 14th, while

the remainder of the water which broke in that day was going down the ventilation raise and thence through the stopes to the 14th levels. Water was also coming out of the Zero section on the 12th.

At the pump stations, the four big triple-expansion flywheel pumps and the other emergency pumps on the 12th level were all running overspeed and were pumping at the rate of 7,000 gal. per minute, but making little impression on the flow.

The 14th level Prescott pump was lifting about 700 gal. per minute to the drain tunnel while the sumps on both 12th and 14th levels were overflowing down to the 16th level.

A record of the happenings during the flood period showed for the first few days a gradual tendency to decrease on the 10th level, a disheartening succession of periodic heavy surges, and rushes from the latest water source on the 12th, and an almost constant overflowing of the sumps on the 12th and 14th levels. The water in the Zero country, which we had been trying to drain for some time previous, behaved erratically for several days, coming in big rushes at times, and then subsiding, but after a few days this source gradually drained and in time almost completely dried up.

Each day regular and very thorough rounds of inspection were made to all the points on the 10th, 12th, 13th, and 14th levels, where there were any water flows, and the conditions carefully noted and reported, and any repair work that was necessary to the drifts done.

The longest and most arduous piece of work was mucking out the 12th level drifts leading from the 12th pump station back to where the latest water had broken in. The difficulties were considerable and of sufficient interest to relate in detail.

The first rush of water had brought down, as noted before, a tremendous volume of débris, ranging in size from boulders 2 ft. in diameter, down through small rocks to fine dacite sand. The heavier and coarser material was deposited in the first few hundred feet of drift away from the "break," often as high as 5 ft. in the drift. The coarser sands were carried farther and completely covered the mine car tracks to a depth varying from 1 to 4 ft., while the fine sands really never settled, and were carried in suspension by the water into the sumps.

To keep the main exit in the water crosscut open entailed the shoveling away of the débris as fast as it accumulated at the mouth of the break in order to prevent choking up. At first, so as to have use of the tracks and be able to use mine cars, it was attempted to start cleaning near the pump station, and work in from there, but this early proved futile, for no sooner had 100 ft. or so been cleared, than the shoveling of the sand farther upstream put more sand into suspension, which, in a short time, settled and again covered the tracks, and it was soon realized that the objective point never would be reached by this method.

In order to make faster headway, two methods were adopted to get rid temporarily of the sand and rocks and keep the drifts open. A large proportion of the drifts leading from the pump station to the water crosscut were originally timbered; 2 by 12 planks were nailed horizontally between the posts of each set on either side of the drift as low down as the water would allow, and additional planks nailed on to the face of the posts, forming a space 12 in. wide, 4 ft. long, and about 4 ft. high, into which the sand could be shoveled. By doing this all along the timbered portions of the drifts, considerable sand and débris was removed at the minimum expense. Where the sand had to be carried downstream some distance before shoveling into these temporary storage places, in place of using shovels, a group of men placed about 10 ft. apart took 2 by 12 planks, 5 ft. long, and by holding them about 1 ft. off the bottom of the drift, the sand was automatically sucked under one plank, carried by the water without settling to the next plank, and so on until it reached the desired point. As this method of disposal was only adaptable to the sandy portion of the débris, shallow dams were placed in the drifts opposite suitable crosscuts and platforms placed on the top of these dams. Boats about 10 ft. long and 1 ft. deep were then constructed. These boats were towed upstream several hundred feet, loaded with rock and then floated downstream on to the platforms where their contents were shoveled into the crosscuts. By dividing the drifts into several sections, in each of which two or three boats operated, good headway under the conditions was made. In fact, the absolute impossibility of keeping the tracks passable for cars made the boat system the only one feasible for transporting the sand and rock any considerable distance.

It was an exceedingly unenviable and disheartening piece of work, carried out under very trying conditions. The water was running 3 and 4 ft. high in the drifts where the work had to be done, so that men were working over their knees and often up to their waists in water, the temperature of which averaged 65° F. Conditions such as these are not conducive to very high efficiency, but the men were at all times cheerful and carried on the work in a meritorious manner that cannot be too highly commended. Hot coffee was brewed on surface, and it was one man's job each shift to keep all of the men working in the water well supplied with hot drinks.

The discouraging feature of this drift-cleaning work was that often hardly had a section of track been cleaned when a fresh rush of sand and water would come surging down the drift and again fill everything up.

Near the pump station, the dams were made more frequent, but at the best the fine sand was carried over into the sumps with the water. The sumps were kept cleaned of sand as much as possible by an air ejector which discharged into a mine car.

The water above the 16th level had risen, from Mar. 4 to 13, to a

distance of about 36 ft. in the "A" shaft above the level of the station. Fig. 10 shows the area which this water occupied at each foot of rise above the level, and the rate at which it was unwatered.

As has been described before, air lifts were used altogether for unwatering this level, but they could only be used when the water being produced on the 12th and 14th levels would allow of the pumps on these levels handling any additional flows, but there were no special difficulties encountered in reaching the 16th, other than the necessarily slow progress that was made.

A description of the first few days of the flood is practically a description of the whole month during which mining operations in the mine

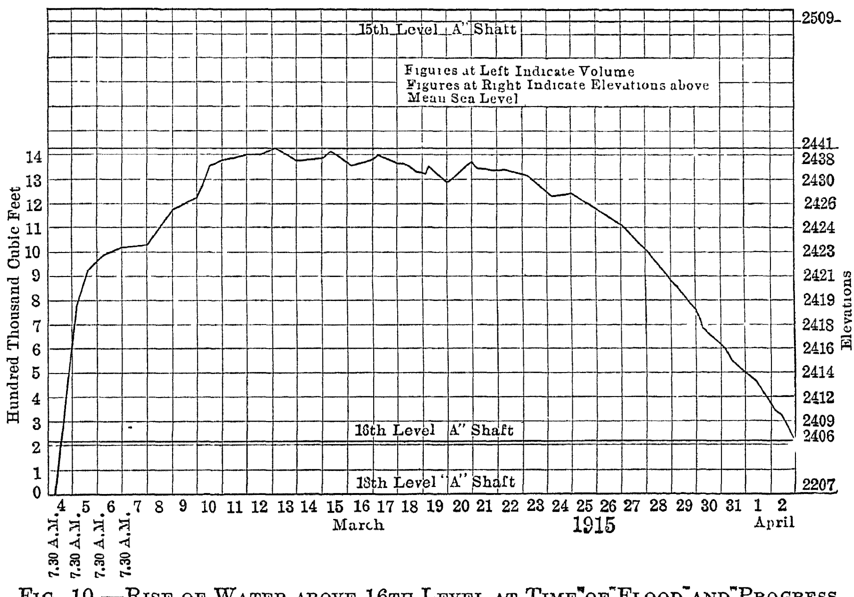


FIG. 10.—RISE OF WATER ABOVE 16TH LEVEL AT TIME OF FLOOD AND PROGRESS MADE IN UNWATERING.

had to be discontinued owing to water troubles. In general, the 10th level water showed a small but steady decrease, and the Zero country on the 12th level became, in time, comparatively dry. The 14th level water from the west country showed a decrease, due more or less to complete choking up with sand of the 13–1–11 raise, and the surges from the new water source on the 12th became less frequent and none were recorded after about Mar. 25. The 16th level was finally unwatered on Apr, 3, or almost exactly a month from the date it was flooded. By the first of April, the flow being handled by the 14th, 12th, and 10th level pumps and air lifts amounted to 12,500,000 gal. per day, and from that time on, the decrease, as is shown in Fig. 6, was comparatively steady but slow. By Apr. 6 the mine was operating again on a small scale, and

steps were being taken to repair the not inconsiderable damage done by the flood.

V. GENERAL CONCLUSIONS

At present (December, 1915), a large flow of water is still running in the channel through which the water broke into the mine, and it is therefore not open for inspection. It is doubtful, even if in time it should turn dry, whether we would be justified in satisfying our curiosity as to the size and general character of the opening at the expense of the risk of tapping additional water, or in making any attempt to block it up to prevent inrushes through the same channel.

With reference particularly to the flow of water from the west side of the mine, judging from our recent experiences and from all the data available over a number of years, indications point to the main mass of water which the mine is draining being held in the diorite formation immediately overlying the dacite. Without doubt this supply of water is fed by the precipitation that falls annually in the district, but from how extensive an area it is impossible to say. During the abnormally wet winter of 1914–1915, it was noticeable that the effect of the yearly portion of the rainfall (up to December) was not felt in the mine. This, doubtless, corresponded to the ordinary season's precipitation, which normally is absorbed by the dry ground and only finds its way into the water basin comparatively slowly. But as soon as the ground was saturated and had absorbed all the moisture possible, all additional rainfall sank and flowed very quickly into the openings in the mine. This, I think, is shown by the flat line of mine water curve up to the end of November, in Fig. 6, when 13 in. of rainfall had fallen since July 1, which period might be called the "saturation" period, and by the appalling rapidity with which heavy rains and floods after that date were felt in the mine. It is possible, in view of the porous nature of the creek bed, that the water flowing continuously in the creek accelerated the seepage into the mine, but with the whole surface area kept in a water-logged condition by the continuous soaking rains, the part played by the creek at this particular time is uncertain. However, as a possible preventive measure, the fluming of the creek over and past the more porous formations, such as the diorite adjacent to the company property and considerably beyond the limit of the old flume, is a logical precaution to take. Although, as noted previously, that portion of the water that the mine is draining on the 12th level is, in all probability, in the diorite, fractures and fissures in the underlying dacite afford easy channels for the downward seepage of the water, and it is in the latter formation that the flows are encountered.

As to conditions in the mine during the period of the rise of water, everything points to the water backing up from the 12th level on the

flat diorite dacite contact, first showing on the 12th, then on the 11th, and then on the 10th levels, and finally in the "C" shaft on the 9th level and above. With insufficient openings for relief, and with practically only one drainage point open on the 10th and one on the 12th level, the pressure became too great and the water broke through at a point on the 12th level connected with a fracture through the dacite where the resistance was weak. It is impossible, of course, to account altogether logically for the water not choosing the more broken up and already wet stoping country in the Zero section, only a few hundred feet away; but probably the zone of the Old Dominion vein, and the system of fracturing in the dacite were the determining factors. With the pressure relieved at the low point on the 12th, the head quickly dropped on the 9th and 10th, but after the first few days following the break, the decrease on the upper levels was very gradual and seemingly not commensurate with the quantity of water pumped from the 12th. This, however, probably only indicates that the filling up of the empty pores of the country was a gradual process carried out on a big scale, and that the complete draining of this same area will be correspondingly slow.

As a preventive measure against uncontrolled flows of water drowning the pump stations, concrete bulkheads in the drifts between the possible sources of water and the pump stations would be of benefit. They should have doors that can be shut tight and fitted with suitable pipes and valves through the bulkheads to allow of no more water coming to the pumps than they can handle. This is essential to avoid the tremendous surges bringing sand and gravel right to the pumps. If properly placed, these would also serve the double purpose of shutting off portions of the mine from the flooded levels, and allowing mining operations to continue in the remainder.

As was natural, the pumps on the 12th level worked at their greatest efficiency prior to Mar. 4, when the flood of water brought sand into the sumps. Although settling ponds were built along the drift, and a final dam near the sump was constructed with a steel screen and burlap covering, it was almost impossible to keep the fine sand away from the suctions of the various pumps, and not only was their efficiency much impaired thereby, but the constant changing of valves caused by sand trouble kept one pump out of commission most of the time, and greatly decreased the total pumping capacity. No valve facing was found that would stand the wear of the sand for long, and the replacing of the big valves on the triple-expansion flywheel pumps, at best a somewhat cumbersome job, was for many weeks a continuous performance.

The valves used in these pumps are 18 in. in diameter and eight in number—four suction and four discharge. The weight of the valve and the valve seat is 300 lb. The valve covers weigh about 600 lb. and are bolted down with 20 1¾-in. bolts. On account of the great size of the valve,

should particles of coarse sand or other substances get under the val holding it up, the sand and water rush through and cut it out v quickly.

Previous to the flood period, we had experimented with numer kinds of valve facings—brass, steel, fiber, and leather. A record of va service on the triple-expansion flywheel pumps shows that with cle water conditions, the brass valves average 37 days in service, the st valves 70, the fiber 105, and the leather 64. Pump No. 2, which is clear-water pump and takes its water direct from the concrete dam, run as long as 180 days without a valve change.

The cost for the various valves are: Brass, $15.37; steel, $3. fiber $17.82; and leather, $3.86, each. The fiber and leather valves ha to be discarded when once worn out, while the brass and steel valves be refaced at a cost of 55 c. and 65 c. respectively. The above figu from normal conditions with these pumps favor the use of the ste faced valve, but with gritty and sandy water the steel valve gave v short service, and leather, fiber, and rubber were all tried in turn. T fiber and hard leather valves were found preferable, the better grades fiber lasting from 4 to 6 days, and hard leather about the same length time.

With the Prescott triple-expansion pumps, the gritty water led to same valve difficulties, but the work of changing the valves, on accou of their smaller size, was infinitely easier. When these latter pumps w running at from 33 to 37 r.p.m. with sandy water, new valves were requi about every 48 hr. in both suction and discharge chambers, and we seen to get better service with the hard rubber valve than with any ot kind. At the normal speed and with clear water, rubber valves also g best service in the triple-expansion Prescott pumps, and are often in go condition at the end of two months.

On account of the valve trouble experienced with high efficien pumps of this type, where there is any likelihood of having sandy water pump, special attention should be paid to the valve question, and wit proper limits more and smaller valves are preferable to fewer and big ones; this, on account of their being easier to change, and also the f that one or two bad valves on a pump where the total number of valve small, lowers the efficiency quicker than the same number of poor val on a pump where the total number is much greater; and loss of efficien is a fatal drawback when so much depends on the amount of water to handled. On the other hand, with the pumps running at high spe greater breakage was experienced with the small valves than with the ones on the flywheel pumps.

If electric pumps are installed for emergency service, they should pro bly be of the centrifugal type, with lower efficiency, but with the abil to pump sand and grit and needing little attention. The attention f

ture is one of the most important in emergency pumping conditions. There seems to be a growing tendency among pump manufacturers to show increased interest in the production of a centrifugal type pump with relatively high efficiency. This is commendable, as the market for such a pump is large.

Where there is no surplus steam or electrical power, and where compressed air is available, air lifts will be found more satisfactory, under the right conditions, than sinking pumps run by air power; also infinitely less trouble is caused, air lifts being more or less automatic in their operation. However, a 30 to 40 per cent. submergence is necessary for fairly efficient work. Although as a pumping unit they are uneconomical, they throw large volumes of water, and can be used to very good purpose under certain conditions. They cannot, of course, owing to the submergence necessary, be used down to the lowest unwatering point on a level.

Recent work in unwatering the 18th level, flooded in March, with a 24-in., 5-stage Layne & Bowler deep-well pump showed that it has advantages, where the space will allow, and is much superior to air- or steam-driven sinking pumps for unwatering purposes.

DISTRIBUTION OF PUMPING EXPENSE

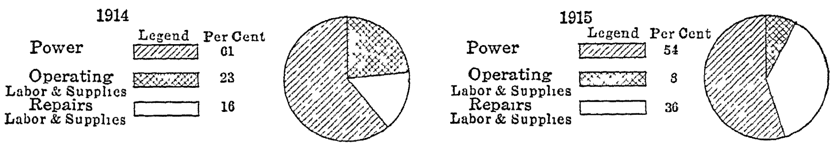


FIG. 11.

Where there is any chance of mud or sand getting into the sumps, care should be taken in the design to allow of their being in duplicate, or at least so partitioned as to admit of thorough cleaning of one section without stirring up the sand in the water delivered to the suctions of the pumps.

Finally, it may be of interest to state some figures on the pumping expense connected with the handling of these large volumes of water. To arrive at an adequate conception of the amount of water pumped from the mine in the first six months of 1915, it is interesting to note that during this period 6,750,000 tons of water was lifted out of the mine from the 10th, 12th, and 14th levels. During this same period, 99,000 tons of ore was hoisted from the mine, or, in other words, for every ton of ore extracted and hoisted, the Old Dominion Copper Co. had to pump out of the mine 68 tons of water, a proportion of water to ore that is, obviously, not conducive to cheap mining. During the first half of the year, the pumping expense amounted to 29 per cent. of the total underground, hoisting, and surface expense. Details of the pumping costs during the flood period were so abnormal as to have no particular interest, but it may be stated that of the total pumping expense, the operating labor and

supplies amounted to 10 per cent., power to 54 per cent., and repair labor and supplies to 36 per cent. In the previous year, 1914, operating labor and supplies amounted to 23 per cent. of the total, power 61 per cent., and repair labor and supplies 16 per cent. This is shown graphically in Fig. 11. The repair expense consists of repairs, labor and material for steam lines, water columns, pumps, sumps, and drain tunnel, as well as the installation of emergency pumps, air lifts, etc. In the power expense, which is the heaviest individual item of the pumping costs, fuel oil is practically 87 per cent. of the total, which, during the period from Jan. 1 to June 30, 1915, would make the fuel oil expense 47 per cent. of the total pumping costs.

In conclusion, my thanks for the compilation of certain portions of the data used in this paper are due to I. H. Barkdoll, Mine Superintendent, H. L. Norton, Chief Engineer, Charles Mendelsohn, Mechanical Engineer, and G. N. Bjorge, Geologist.

The Composition of the Rock Gas of the Cripple Creek Mining District, Colorado*

BY GEORGE A. BURRELL† AND ALFRED W. GAUGER, PITTSBURGH, PA.

(Arizona Meeting, September, 1916)

Introduction

THE senior author of this paper, while in Colorado on other official business, made a trip to the Cripple Creek gold-mining district to get more data than are at present available regarding the composition of the gas that issues from the rock into the mines of the district. This was done at the suggestion of George S. Rice, Chief Mining Engineer of the Bureau of Mines. There are many questions of interest and importance in connection with the ventilation of the mines in the Cripple Creek district, such as the menace to life and mining of a suffocating gas entering the mines from the rock, the danger due to powder smoke (from blasting operations), and danger of continued inhalation of rock dust by the miners. This report confines itself to the first problem, the gas from the rock, and principally to one phase of this question, the composition of the gas and its effects on men and lights. A more thorough study of this and other ventilation problems is planned.

That the rock gas is a real menace is shown by the fact that miners (variously estimated from 25 to 50 by different persons interviewed) have been killed by it in the 25 years that mining has been vigorously carried on at Cripple Creek. Many men have had narrow escapes from death, some of them having been incapacitated for days. In addition, much loss of time results because at certain periods of the year or on certain days it is impossible to enter some workings. At a few mines, fans force air into the workings, thus supplementing the natural ventilation and much improving conditions, but the improvement is not entirely adequate at all times.

Origin of the Gas

Lindgren and Ransome[1] think that the gas found in the rocks of the Cripple Creek mining district represents the last exhalations of the ex-

* Published by permission of the Director of the Bureau of Mines.

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[1] W. Lindgren and F. L. Ransome: Geology and Gold Deposits of the Cripple Creek Mining District, *Professional Paper* 54, *U. S. Geological Survey*, p. 257 (1906).

tinct Cripple Creek volcano. In support of this they point out that little timbering is used in the mines, hence one cannot account for the decrease of oxygen or increase of carbon dioxide by oxidation of timber. Further, only a small proportion of pyrite and carbonates is present in the rocks and ores as compared to that in many other mines entirely free of gas. Moreover, no gas occurs in the oxidized zone, showing that oxidation cannot have anything to do with it, and since the gas increased with depth it must be mainly accumulated below the present workings.

A theory other than that propounded by Lindgren and Ransome attempts to account for the occurrence of the gas because of intrusion of atmospheric air and the removal of the oxygen by means of oxidation and perhaps to some extent by absorption due to underground waters.

Influence of Barometric Pressure on the Outflow of Gas

The gas is confined in the rock under very low pressure, so low, in fact, that variations in the outside atmospheric pressure materially affect the outflow of gas into the mines. That the gas is in the rocks under low pressure is shown by the fact that at the few mines where air is forced in by fans and the workings are thus under about 6 or 7 in. of water pressure (about 0.5 in. of mercury), the gas is forced back into the pores and crevices of the rock and the workings are more or less free of it. The variation in outside atmospheric pressure is frequently in excess of 0.5 in. of mercury, so that largely depending on this variation the air of some of the mines is more or less contaminated with the rock gas. This influence of the barometric pressure on the outflow of gas is appreciated by the mining men of the district.

Table I.—*Data of Samples Collected near Feeder in Mary McKinney Mine*

Sample Number	Date of Collection of Sample	Analysis			Direction of Wind	Barometric Pressure Inches, Hg	Condition of Weather	Strength of Wind
		CO_2	O_2	N_2				
6987	Nov. 1	0 24	20.58	79.18	N.-N.W.	22.25	Clear	Mild
6984	Nov. 3	12.06	2 97	84.97	S.W.	22.26	Clear	Mild
6986	Nov. 5	13.27	0.94	85.79	S.-S.W.	22.16	Partly cloudy	Mild
6981	Nov. 9	8.84	7.83	83.33	S.-S.W.	22.06	Partly cloudy	Mild
6983	Nov. 11	0.17	20.75	79.08	N.W -N.	21.96	Clear and cold	Strong
6980	Nov. 13	11.05	3 39	85 56	S.-S.W.	21.93	Partly cloudy	Mild
6988	Nov. 15	7.86	8.89	83.25	N.W.-N.W.	22.00	Clear and cold	Strong
6989	Nov. 22	0.41	20.64	78.95	N.-N.W.	22.16	Clear	Mild

A number of samples were collected at the instance of the authors of this paper, by A. G. Suydam, mining engineer, Cripple Creek, Col.

TABLE II.—*Influence of Wind Direction on Strata Gas*

Date, 1915	Barometer	Wind		Sky	Remarks
		Direction	Strength		
Oct. 11	21 875	S.-S.W.	Mild	Cloudy	Mine closed because of strata gas.
Oct 12	21.955	N.-N E.	Calm	Clear	Air good in drift.
Oct. 14	21 985	N.E.-N.	Calm	Cloudy	Some strata gas in drift.
Oct. 15	22 030	N.E.-N.	Strong	Cloudy	Air good.
Oct. 16	22.130	S.W.-S.	Calm	Cloudy	Air good.
Oct. 17	22.185	N.-N.W.	Mild	Clear	Air good.
Oct. 19	22.175	N.-N.W.	Mild	Clear	Air good.
Oct. 21	22 240	S.-S.E.	Mild	Clear	Air good.
Oct. 22	22.350	S.-S.E.	Calm	Clear	Some strata gas in drift.
Oct. 23	22 295	S.-S.E.	Calm	Clear	Some strata gas in drift.
Oct. 25	22.200	N.-N.W.	Calm	Clear	Air good.
Oct. 26	22 290	N.-N.E.	Mild	Clear	Air good.
Oct. 27	22.380	N.-N.W.	Calm	Clear	Air good.
Oct. 28	22 385	N.-N.W.	Calm	Clear	Air good.
Oct. 30	22 235	S.W.-S.-W.	Strong	Partly cloudy	Strata gas in drift.
Nov. 2	22 260	W.-S.W.-N.W.	Mild	Partly cloudy	Air good.
Nov. 10	21.750	S -S.W.	Mild	Fog	Air good.
Nov. 16	21.800	N.-N.W.	Mild	Clear (cold)	Strata gas in drift.
Nov. 18	. ...	N.-N.W.	Mild	Clear (cold)	Air good.
Nov. 19	22.185	N.W.-N.	Mild	Clear	Air good.

They were forwarded to the Laboratory of the Bureau of Mines for analysis. The samples were collected almost daily for a period extending over 22 days, in the Mary McKinney mine in the No. 12 North Drift, 800-ft. level about 1,800 ft. from the shaft. The point of sampling was very close to a fissure in the rock from which a "feeder" of strata gas intermittently issued. Table I shows the number of the samples, date of sampling, the analyses, and the direction of the wind and barometric pressure on the date the samples were collected.

In addition to the results in Table I, some observations were made by Mr. Suydam in connection with which gas samples were not collected. The condition of the air in the drift was tabulated according as it affected candles and acetylene lights. These data are shown in Table II.

Comments on Results

Referring first to those samples that were collected and analyzed (Table I): The total number of samples collected was too small to draw rigid conclusions as regards a connection between the barometric pressure and amount of strata gas as shown by the carbon dioxide and oxygen

percentages. In this very short series of tests the amount of strata gas is not a function of a low barometric pressure. With one exception (sample No. 6988) the largest amounts of carbon dioxide and smallest amounts of oxygen were found in samples collected when the wind was from a southern or southwestern direction. This coincides with the experience of some mining men in the Cripple Creek district who informed the senior author that it was frequently true that when the wind came from a southern or southwestern direction the most strata gas entered the mines.

As regards Table II, it is shown that strata gas was present in the north drift of the Mary McKinney mine when the wind direction was southern or southwestern in four cases and northerly in one case. Also, in 10 cases the air was good in the drift when the wind came from a northerly direction, and in three cases when it came from a south or southwestern direction. Again a consistent relation could not be traced between the barometric pressure and the presence of strata gas.

Some of the mining men of the Cripple Creek mining district believe that the direction of the wind influences the outbursts of strata gas, in that when it comes from a southern or southwestern direction it sweeps up the cañons. The rock being more or less porous, permits the entrance of atmospheric air, and this air, exerting a pressure on the rock gas, forces the latter into the mine workings.

Collection of Gas Samples

Samples of gas were collected by the senior author in the Mary McKinney, Anaconda, Midget, and Cresson mines. The Mary McKinney mine is on the south side of Squaw Gulch, opposite the town of Anaconda. The first ore was shipped in 1893. The workings are mainly in breccia and phonolite.

The Anaconda mine is in the town of Anaconda, with workings extending through Gold Hill, and was one of the first producers in the Anaconda district. The mine is at present worked through an adit having a portal at Anaconda and connecting with extensive drifts and crosscuts. The prevailing country rock is breccia of the usual type found on Gold Hill. Within the breccia are some irregular bodies of latite-phonolite and a few dikes of phonolite and basalt.

The Midget mine is on the west slope of Gold Hill. The workings comprise 10 levels and are partly in breccia and partly in fine-grained gray gneiss.

The Cresson mine is on Raven Hill, 2½ miles from the Midget mine and 2 miles from the Anaconda mine. The workings are in breccia. The mine was opened in 1904 and has 13 levels.

TABLE III.—*Rock-Gas Analyses*
Midget Mine

Sample Number	CO_2	O_2	Combustible Gas	N_2	Total
781	3.66	16.57	0.00	79.77	100 00
776	1.00	18.92	0 00	80 08	100.00
663	0.93	20.03	0.00	79.04	100.00
770	8.84	10.86	Trace[a]	80 30	100.00
296	2.08	18.53	0.00	79.39	100.00
769	0.11	20.79	0.00	78.70	100.00
400	8.68	10.86	0.00	80.46	100.00
794	7.35	11.63	0.00	81.02	100.00
664	5.09	15.06	0.00	79.85	100.00

[a] Less than 0.02 per cent.

MIDGET MINE

General Observations Regarding Gaseous Conditions

The Midget mine is on Gold Hill and has 10 levels. From the shaft to the tenth level is about 900 ft. The ore veins range from 5 in. to 30 ft., probably averaging 4 ft. About 20 men work underground on a leasing system. Regarding the levels where samples of gas were collected: Four men were working in the seventh level, four men on the eighth level, and none, on account of bad gas conditions, on the ninth or tenth levels.

In the afternoon of the day when the above samples were collected the men had to leave the eighth level because gas conditions became so bad that working was impossible. Gas conditions were said to be worse than usual at the Midget mine that day.

The Midget mine management has in use a pressure system of ventilation. A 5-ft. pressure Sturtevant fan driven by a 20-hp. motor is installed at the surface, and an air compartment built in the shaft conducts the fresh outside air to a point below the second level; from this place the air spreads in the shaft and into the various levels. The drifts of the latter at various distances from the shaft are provided with air doors, built of 1-in. boards and made as air-tight as possible with canvas. Thus, a small pressure is placed on the mine workings, sufficient to check in part the outflow of gas from the rock strata. This pressure amounts in some places to 5 or 6 in. of water. The fact that the gas is in the rock at very low pressure makes such an arrangement efficient in that much more work can be accomplished by the men because of the better conditions of ventilation. Even so, however, when the outflow of gas is stronger than usual, this method is not entirely successful, in that workmen are sometimes driven from the lower levels, and sometimes cannot re-enter for days.

Description of Samples Collected

Sample 781.—Sample collected in the eighth level, north drift; Midget Murray vein. Wet bulb 73° F., dry bulb 76° F.; relative humidity, 88 per cent.; barometer, 21.6 in. Candle would not burn where sample was collected. Acetylene lamp did burn. No men were working at this place.

Sample 776.—Collected in eighth level, in the Olson Hollengrain stope. Candle and acetylene lamp burned. Two men were working where sample was collected. Compressed air had been turned on in this stope, through a 1-in. line, for 2 hr. prior to taking the sample. The end of the air line was 50 ft. from the stope. The top of the stope lay at an angle of 45° from the end of the air line.

Sample 663.—Sample collected in eighth level at breast. Lauson and Haug's lease on the south drift. Midget Murray vein. Two men working in this breast when sample was collected. Wet bulb 60° F., dry bulb 65° F.; relative humidity, 75 per cent. Candle and acetylene lamp would burn. Compressed air had been turned on in this breast for 2 hr. prior to taking sample. The breast was 75 ft. from the end of the air line.

Sample 770.—Sample collected in raise on Intermediate vein 40 ft. from breast, eighth level. No compressed air in the place. Candle and acetylene lamp went out but the latter would burn 1 ft. above where sample was collected. Wet bulb 63.5° F., dry bulb 69° F.; relative humidity, 76 per cent. Nobody working in this place.

Sample 296.—Collected in ninth level. Conundrum vein. North drift, 250 ft. from shaft. Wet bulb 63° F., dry bulb 69° F.; relative humidity, 76 per cent.; barometer, 21.66 in. Candle burned feebly at place where sample was collected. No men working on ninth level.

Sample 769.—Sample collected in the shaft at ninth level, 830 ft. from the surface. Wet bulb 61° F., dry bulb 66° F., relative humidity 74 per cent. Air in the shaft would not move the vanes of the anemometer, although there was a slight movement of air downward due to the fact that a 5-ft. Sturtevant fan at the top of the shaft was forcing air into the mine.

Sample 400.—Sample collected in seventh level. Intermediate vein, 375 ft. northwest of shaft. Carbide lamp went out where sample was collected, but burned 2 ft. above. No men working here. Wet bulb 63.5° F., dry bulb 74° F.; relative humidity, 59 per cent.

Sample 794.—Collected on seventh level, Midget Murray vein, 250 ft. north of shaft and 75 ft. from breast. Carbide lamp went out where sample was collected. Nobody working here.

Sample 664.—Collected on sixth level. Outside of pressure zone. About 200 ft. beyond air door. Wet bulb 50.5° F., dry bulb 54° F.; relative humidity, 81 per cent.

Of the series of samples collected at the Midget mine, only one, No. 664, was collected outside the pressure area or zone and beyond the door. Candles burned all right inside the door, but 20 ft. beyond they went out, indicating at this point less than about 17 per cent. of oxygen. That considerable pressure was exerted on the doors was shown by the fact that it required much force to open them and that as one did so the air rushed through sufficiently strong to blow out the lights. This rush of comparatively fresh air through the door and beyond into the drift, of course, improved conditions there so that up to 20 ft. the oxygen content was sufficiently high (17 per cent.) to support the flame of a candle.

It should be added that the management of the Conundrum mine was the first in the Cripple Creek to install this so-called pressure system of ventilation. This mine adjoins the Midget, and after that installation, the rock gas was forced in greater quantity into the Midget mine, hence a similar system had to be installed in the latter mine for self-protection.

TABLE IV.—*Rock-Gas Analyses*

Anaconda Mine

Sample Number	CO_2	O_2	Combustible Gas	N_2	Total
691	9.02	5.51	Trace[a]	85 47	100 00
966	7 96	7.50	0 02	84.52	100 00
937	5 43	11.78	0.00	92.79	100 00
750	8.09	7.19	Trace[a]	84.72	100.00
694	1.64	18.30	0.00	80.06	100 00
948	1.51	18.44	0 00	80.05	100.00
690	1.69	17.70	0.00	80.61	100.00
747	0.32	20.46	0.00	79.22	100.00

[a] Less than 0.02 per cent.

ANACONDA MINE

Description of Samples Collected

Sample 691.—Collected in fourth level, fourth drift, at floor 300 ft. north of shaft. Carbide lamp went out at roof and floor, but burned at about the position of a man's head when walking upright. Wet bulb 56° F., dry bulb 59° F.; relative humidity, 85 per cent.

Sample 966.—Collected in Anaconda mine, fourth level, fourth drift, 275 ft. from shaft. Wet bulb 56° F., dry bulb 59° F.; relative humidity, 85 per cent.; barometer 21.6 in. Nobody was working in this level this day on account of the bad condition of the air. It is almost always the case that the influx of the gas prevents wórking on the fourth level in the afternoons, and about one-third the time in the morning.

Sample 937.—This sample was collected in the same drift, same level, except about 250 ft. from the shaft. Sample collected in drift at

about the place the carbide lamp was just extinguished. The percentage of oxygen, it will be noted, is 11.78, and corresponds with experiments in the senior author's laboratory which have shown that the acetylene lamp is extinguished when the oxygen content of an atmosphere drops to about 12 to 13 per cent.

Sample 750.—Sample collected in same level and same drift as above samples except 150 ft. from shaft. Sample collected near the floor. Carbide lamp went out near the floor.

Sample 948.—Same as above except taken at height of man's head when walking upright. Candle burned dimly.

Sample 694.—Same as above except taken at roof, 7 ft. from floor.

Sample 690.—Taken in fifth level at breast of north drift. This level is 200 ft. below fourth level. Candle would not burn but carbide lamp would. Two men were working here. Compressed air had been turned on in this breast up to 3 min. before sample was collected. Wet bulb 64° F., dry bulb 68° F.; relative humidity, 82 per cent.

Sample 747.—Taken at foot of shaft on fifth level. Air was being forced into this level from a fan on the fourth level. Candle would burn. Wet bulb 57.5° F., dry bulb 60.5° F.; relative humidity, 85 per cent.

General Observations Regarding Gaseous Conditions.—The Anaconda shaft is reached by means of an adit 1,135 ft. long. The fourth, fifth and lower levels are at times badly affected by outflow of strata gas.

The progressive vitiation of the air in the fourth drift of the fourth level is shown by the first six samples. Sample 691 taken 300 ft. north of the shaft contained 9.02 per cent. of carbon dioxide and 5.51 per cent. of oxygen. In taking this sample, the sampler proceeded with the acetylene lamp held near his mouth, until the lamp went out (about 12 or 13 per cent. oxygen), then reached down and collected the sample near the floor. The dangerous stratification of the gas is well brought out by these analyses. At the height of a man's mouth the air was breathable, but if a man fell by some mishap into the air near the floor, collapse would quickly occur because of the small amount of oxygen (5.51 per cent.) there. At a point 25 ft. nearer the shaft (sample 966) at the floor the air was a trifle better: 7.96 per cent. carbon dioxide and 7.50 per cent. oxygen. At a point 50 ft. nearer the shaft (sample 937), near the floor the air was still better: 5.43 per cent. of carbon dioxide and 11.78 per cent. of oxygen. The next three samples were collected 150 ft. from the shaft: one at floor (No. 750), one at the height of a man's mouth (No. 948) and one near the roof (No. 694). At the floor a very dangerous atmosphere existed, while nearer the roof the air was comparatively good. That the atmosphere in this drift, as in other mines, varied much in composition is shown by samples 937 and 750. Sample 937 was taken at a point 250 ft. from the shaft. This sample contained 11.78 per cent. of oxygen, whereas sample 750 collected closer

to the source of fresh air (150 ft. from the shaft) and farther away from the most contaminated air, contained a fatal proportion of oxygen, 7.19 per cent.

It was noticed on another occasion that a lighted candle would burn fairly well at one point in a drift, then become extinguished farther along (great care being taken that it was not a sudden movement or jerk of the candle that put the flame out) and finally would stay lighted at a point still farther in the drift.

Usually, though, the nearer one approached the breast of a drift containing much rock gas the worse the air became.

TABLE V.—*Rock-Gas Analyses*

Mary McKinney Mine

Sample Number	CO_2	O_2	Combustible Gas	N_2
761	1.82	10.70	0.00	87.48
762	7.54	8 01	0.00	84.45
780	4.17	13 87	Trace[a]	81.96
908	0.25	20.50	0 02	79.23
976	5.88	10.71	0.00	83.41

[a] Less than 0.02 per cent.

MARY MCKINNEY MINE

Description of Samples

Sample 761.—Collected on eighth level (about 800 ft. from surface), No. 15 raise on No. 2 vein, up about 15 ft. from level of drift and about 1,400 ft. from shaft. The air was worse the higher up one went in the raise. The sample was collected about 10 ft. up the raise from where the carbide lamp went out. Nobody was working in this raise. Wet bulb 56° F., dry bulb 60° F.; relative humidity, 80 per cent., barometer 21.87.

Sample 762.—Collected in eighth level, at floor 180 ft. north of black vein, No. 12 north drift, and 1,500 ft. from shaft. Carbide lamp burned at roof but would not burn at floor, where sample was collected. Nobody working at this place. The gas sometimes fills the entire drift. Conditions were worse than usual on the day samples were collected. Wet bulb 58.5° F., dry bulb 61° F.; relative humidity, 88 per cent.

Sample 780.—Sample taken in eighth level at roof. No. 12 north drift at junction of black vein drift. Carbide lamp burned.

Sample 908.—Sample taken at junction of No. 2 vein and main crosscut from shaft, 1,250 ft. from shaft. The shaft was acting as a downcast and this sample was collected 250 ft. from the upcast. Wet bulb 54.5° F., dry bulb 58° F.; relative humidity, 82 per cent. Air seemed good at this point. There was a slight movement of air (14 lin. ft. per minute).

TABLE VI.—*Rock-Gas Analyses*

Cresson Mine

Sample Number	CO_2	O_2	Combustible Gas	N_2
797	8.90	7.03	0.00	84.57
795	2 85	16 23	0.00	80.92
760	11.08	2.69	0.03	86.20
759	2 35	17.19	0.00	80.46
771	0 29	20.56	0 00	79.15
754	1 01	14 05	Trace[a]	80 94
753	0 60	16 74	0 00	82.66
772	0 62	16 65	0.00	82.73
943	0 39	20 03	0 00	79 58
944	0 51	17 78	0.00	81 71

[a] Less than 0.02 per cent.

CRESSON MINE

Different Ventilation System

At this mine the plan of pressure ventilation adopted is slightly different from that at the Midget and Conundrum mines. Wooden doors, made as tight as possible, are placed in the drifts beyond the workings, and a compressed-air pipe is run through a hole therein. A valve placed inside the door provides for the control of the air. With the air turned on, rock gas that would otherwise find its way into the workings is kept back. That this method is capable of doing effective work is shown by the analyses of several samples collected inside and beyond one of these doors.

Description of Samples

Sample 797.—Collected at the floor in the eleventh level at one of the pressure doors 500 ft. from the shaft. The door was opened slightly when the sample was collected. The carbide lamp would not burn at the point where the sample was collected. Dry bulb 69° F.

Sample 795.—Collected in the same place as the above except at the height of a man's head, standing upright. Carbide lamp would burn here.

Sample 760.—Collected beyond the air door, about 10 ft. beyond where the two previous samples were taken. In collecting this sample the collector advanced 10 ft. beyond where the air was comparatively good, holding his breath, then quickly snapped the sealed and evacuated glass sample bottle, whereupon the mine gas immediately filled the bottle, after which he quickly came out. Even so he had a close call, for his knees became weak and mind slightly hazy, due no doubt to his inadvertently breathing a little of the atmosphere. As soon as he came to better air 10 ft. away, he felt all right.

This sample is interesting from two standpoints: (1) It shows the effect of pressure produced by the compressed air in holding back the strata gas. At the height of a man's head just at the door the oxygen content of the atmosphere was 16.23 per cent., while 10 ft. beyond the door the oxygen content was only 2.69 per cent.; (2) this sample more closely approximated the pure rock gas than any other sample collected So-called blowers of gas or pronounced gas feeders, while not unknown, are difficult to find in the Cripple Creek mines. The gas usually escapes from the rocks from thousands of small pores therein, rather than by large outbursts, such as are often encountered in coal mines. Although these gas blowers have been encountered in the mines of the district, none were observed in the course of collecting the samples. If one calculates this sample air-free (No. 760) the composition (by eliminating the nitrogen and oxygen according to the proportions they are found in atmospheric air) becomes 12.69 per cent. carbon dioxide and 87.31 per cent. nitrogen. These figures, then, show the composition of the pure gas as it came from the rocks, on the assumption that the air present in the sample as collected was due to dilution of the gas by the air of the mine.

As regards occasional small outbursts of rock gas, an incident related by William Allen, Superintendent of the Midget mine, is of interest. While leaning against the wall of a drift with his face quite close to the rock, talking to another man, Mr. Allen suddenly felt dizzy, breathless, and weak. Without knowing the cause he changed his position and soon felt better. Thinking that possibly there was a small feeder of gas close to where he had been standing, he put his carbide lamp up against the rock there and the lamp was immediately extinguished.

Sample 759.—Collected in the eleventh level under No. 1,103 stope, 400 ft. from the shaft, *i.e.*, 100 ft. inside the door where the previous three samples were collected. Wet bulb 60° F., dry bulb 64° F. A candle burned well at this place and the air seemed good.

Sample 771.—This sample was collected in the eleventh level at the shaft. The latter was upcasting at the time. Wet bulb 64.5° F., dry bulb 65.3° F.; relative humidity, 81 per cent.

In another drift in this level that ordinarily contained a large amount of rock gas, there seemed to be little on the day of the visit, as one could walk with lighted candles clear to the breast. This condition, the presence, on the same day, of much rock gas in some drifts of a certain level and very little in other drifts that are often filled with it, sometimes prevails.

Samples 754, 753 and 772 were collected in the seventh level, 700 ft. from the shaft. Beyond this point of sampling one could not venture very far because the drift for a distance of 1,000 ft. to the breast was filled with rock gas. Candles would not burn where the samples were

collected but carbide lamps would. Sample 772 was collected at the center of the drift, No. 753 at the top, and 754 near the bottom. The air at the bottom contained more gas (less oxygen) than the air at the center or top. Wet bulb 61° F., dry bulb 64° F.; relative humidity, 86 per cent.

Sample 944.—Collected 50 ft. nearer the shaft than the above three samples.

Sample 943.—Collected 50 ft. nearer the shaft than sample 944. In other words, the atmosphere became progressively better the nearer one approached the shaft.

Composition of the Strata Gas as It Issues from the Rocks in the Cripple Creek Mines

Samples of pure rock gas as it issued from the rocks in the Cripple Creek mines could not be obtained by the senior author at the time of his visit. Some of the drifts most affected were penetrated as far as possible, and a sample containing 2.69 per cent. oxygen (sample 760, Cresson mine) which showed the largest percentage of rock gas was obtained. Undoubtedly, 15 or 20 ft. farther in the drift a sample practically devoid of oxygen would have been secured. One can determine quite closely the composition of the rock gas, however, by selecting those samples that contain the smallest percentages of oxygen and calculating them air-free. In Table VII this has been done.

These recalculations, based on the assumption that the gas issuing

Table VII.—*Rock-Gas Analyses, Cripple Creek Mines with Computation on Air-Free Basis*

Sample Number	CO_2	O_2	Combustible Gas	N_2	Total	Calculated Air-Free CO_2	Calculated Air-Free N_2
			Midget Mine				
770	8 84	10 86	0.00	80.30	100 00	18.37	81.63
400	8 68	10 86	0.00	80.46	100.00	18.03	81.97
794	7.35	11.63	0.00	81.02	100 00	16.53	83.47
			Anaconda Mine				
966	7.96	7 50	0.00	84.54	100.00	12 40	87.60
937	5.43	11 78	0.00	82.79	100 00	12.41	87.59
691	9 02	5 51	0.00	85.47	100.00	12.23	87.77
750	8.09	7 19	0.00	84.72	100.00	12.32	87.68
			Mary McKinney Mine				
762	7 54	8 01	0.00	84.45	100 00	12 20	87.80
976	5 88	10 71	0.00	83.41	100.00	12 03	87.97
			Cresson Mine				
797	8.90	7.03	0.00	84.07	100.00	13.40	86.60
760	11.08	2 69	0.00	86.23	100.00	12 71	87.29

from the rock contains no air, show that the rock gas contains between 12.03 and 18.37 per cent. of carbon dioxide and 81.63 and 87.97 per cent. of nitrogen, the average of all of the results being 13.87 per cent. of carbon dioxide and 86.13 per cent. of nitrogen, or in round numbers 14 per cent. carbon dioxide and 86 per cent. nitrogen. The rock gas then is a mixture of carbon dioxide and nitrogen. The assumption that the oxygen in the samples came from the mine air appears to be justified by the decreasing proportion of oxygen found on advancing from good air into a drift more or less filled with the rock gas.

The senior author noticed a tendency among the mining men of Cripple Creek to speak of the rock gas as carbon dioxide or at least as if carbon dioxide were the predominating constituent. This is not the case. Nitrogen is much in excess. The bad effects produced are principally due to the fact that the rock gas so dilutes the air of the mines that the oxygen therein falls to a point where lights will not burn or life is endangered.

In the author's opinion the acetylene light should not be used as the sole warning against the presence of gas in these mines. It is true that where the acetylene lamp burns there is enough oxygen in the air to support life (12 to 13 per cent.) but the trouble is that under such conditions the air only a short distance beyond in a drift or at the floor may be fatal to life. The warning of a candle flame affords a much wider margin of safety. At some mines the management allows no work to be performed where a candle will not burn. At other mines the men are not so careful.

Effect of Carbon Dioxide on Man

Carbon dioxide must be present in air in large quantities before it threatens life. Symptoms of distress in men usually do not begin to appear until from 3 to 4 per cent. is present. Men can go on working for a considerable time in such an atmosphere although they will certainly become more quickly fatigued. Great exertion will cause panting, but animals have been kept for weeks in such atmospheres without causing them much inconvenience. A percentage of 7 to 8 causes more urgent symptoms while with 10 per cent. the distress is very great, the headache becomes more severe, there is heavy panting, and throbbing and flushing of the face, and the gas begins to have a stupefying effect. With 12 to 15 per cent. cerebral symptoms appear and the patient soon becomes unconscious.[2] Death may take place after exposure for several hours to 25 per cent., but a much greater percentage, over 50 per cent., may be breathed in the case of some animals without causing distress.

[2] The above assumes that the oxygen content is not low enough itself to cause distress.

The effect of breathing air containing carbon dioxide is to increase the depth of breathing. The so-called respiratory center in the brain is stimulated. This is brought about independently of the will of the subject and is caused by the smallest increase in the percentage of carbon dioxide in the inspired air. So small an amount as 2 per cent. of carbon dioxide in the atmosphere is not so much a matter of safety and comfort to those who breath it as it is of their efficiency as workmen. In addition to the work a man may be doing in such an atmosphere he is handicapped to just this extent: He is forced to breathe a larger volume of air in a given time, a feat that consumes energy just as his work of drilling a hole in the rock or loading out ore consumes energy.

In England the law requires that the carbon dioxide content shall not rise above 1.25 per cent. in any part of the mine.[3]

Effect on Man of Atmospheres Low in Oxygen

Abnormally rapid breathing (hyperpnœa) is brought about when men or animals are rapidly subjected to atmospheres low in oxygen even during rest. In some individuals this happens at altitudes about or slightly higher than sea level (normal atmospheric pressure) when the oxygen drops to 13 per cent. However, if the transition from plenty of oxygen to want of oxygen is gradual, the temporary marked hyperpnœa is not noticed.[4] But there is this important difference between the deep breathing caused by excess of carbon dioxide and that caused by low oxygen. When deep breathing is due to excess of carbon dioxide a man is probably in no immediate danger, but when it is caused by low oxygen the danger is usually imminent. In the case of the gases in the mines of the Cripple Creek district, when the oxygen falls low enough to be near the danger point, there is invariably present enough carbon dioxide also to affect the breathing.

Paul Bert[5] showed that the abnormal symptoms and dangers associated with low-oxygen atmospheres depend on the diminished partial pressure of the oxygen. He found that at ordinary atmospheric pressure a cat died when the proportion of oxygen was reduced to about 4.5 per cent. When the pressure was ½ atmosphere the cat died when the percentage of oxygen was 9, and at 2 atmospheres pressure it died when the oxygen percentage was 2.5. At Cripple Creek with about ⅔ of an atmosphere (21 in. barometric) pressure, the cat would have died at 6.75 per cent. of oxygen.

Haldane[6] observes that when the oxygen content of the atmosphere

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[3] Coal Mines Act, Great Britain, 1911.

[4] Haldane and Poulton: *Journal of Physiology*, vol. 37, p. 390 (1908).

[5] *La Pression Barométrique*, 1878.

[6] J. S. Haldane: The Causes of Deaths in Colliery Explosions and Underground Fires, *Report to the Secretary of State for the Home Department*, p. 15 (1896).

is gradually reduced by the absorption of oxygen, or by the addition of nitrogen, very little may be noticed before the occurrence of impairment of the senses and loss of power over the limbs. If the reduction is gradual and the symptoms are carefully watched it will be noticed that at about 12 per cent. of oxygen, that is with a reduction of 9 per cent., the respirations become just perceptibly deeper. At 10 per cent. the respirations are distinctly deeper and more frequent and the lips become slightly bluish. At 8 per cent. the face begins to assume a leaden color, though the distress is still not great. With 5 or 6 per cent. there is marked panting and this is accompanied by clouding of the senses and loss of power over the limbs which would end sooner or later in death.

In a test to determine the effects on man of atmospheres low in oxygen, a member of the Bureau of Mines breathed air in and out of a bag having a capacity of 70 liters, the exhaled carbon dioxide being removed by means of a can of caustic potash inserted between his mouth and the bag, and was rendered unconscious when the oxygen content of the air had fallen to about 7 per cent. The effect of the lack of oxygen is instructive. The subject felt warning symptoms previous to collapse but did not believe himself in serious danger of losing consciousness, and, in fact, wanted to continue the experiment. After recovery, which only required a few seconds, he felt no real distress until some time after the experiment, but on the next day was decidedly unwell.

Pure dry air, as analyzed by volume, contains 20.93 per cent. of oxygen, 0.03 per cent. of carbon dioxide, and 79.04 per cent. of nitrogen. Included in the nitrogen content are the four inactive gases, argon, krypton, neon, and zenon. These, with the exception of argon which constitutes 0.94 per cent. of air, are present in exceedingly small amount.

These gases expand practically the same under the same conditions of temperature and pressure, hence the proportion by volume of oxygen, nitrogen, etc., in air remains the same, no matter what the altitude, *i.e.*, at Cripple Creek, altitude 10,000 ft. above sea level, the percentage by volume of the different constituents in the atmosphere is the same as at sea level, but the weight of the constituents in a given volume of the air changes. At Cripple Creek, barometric pressure 21 in., the weight of oxygen in a cubic foot is only two-thirds of what it is at sea level.

Since it is the partial pressure of oxygen in air that is important as regards the effects on man of atmospheres low in oxygen, people at Cripple Creek should not be able to withstand atmospheres as low in oxygen as people at sea level, volume basis. This is aside from the fact that they are acclimatized to low-oxygen atmospheres. The pressure of the atmosphere at Cripple Creek is two-thirds of the normal atmosphere. This means, then, that if air containing 7.5 per cent. of oxygen at sea level causes a man to collapse he would collapse at Cripple Creek, other

conditions of experiment being identical, in air containing $\frac{7.5}{2/3} = 11.25$ per cent. oxygen, for both of these atmospheres would be practically identical as regards partial pressure of the oxygen. However, people become acclimatized to low oxygen atmospheres and live at altitudes much higher than Cripple Creek.

The Effect of the Gases on the Miners at Cripple Creek

The effects of working in the bad air of the Cripple Creek mines are typical of those produced by atmospheres low in oxygen and high in carbon dioxide. After a day's work, depending of course on the vitiation of the air, the men have a feeling of oppression, heaviness, and lassitude, or sleepiness, loss of appetite, with a general feeling of "not being themselves." When the air gets very bad, say where a candle will not burn, slight exertion causes breathlessness. Much exposure in bad air brings on headaches and nausea and complete exhaustion. That more fatalities do not occur is due to the fact that the men pretty well appreciate the warning of their lamps and are very careful about venturing where acetylene lamps will not burn. Collapse may be very sudden in atmospheres low in oxygen. In fact, it is typical of such atmospheres that they give little warning of grave danger. In some cases men who have collapsed and been rescued have been days recovering. The after-effects are very similar to poisoning by carbon monoxide.

Effect of Reduced Oxygen and Increased Carbon Dioxide on Lights

As the oxygen in air decreases the illuminating power of lights diminishes, until in the case of the ordinary oil-fed lamp wick such as a candle or miner's oil torch, the flame becomes extinguished in air containing 17 per cent. of oxygen. J. S. Haldane[7] observed that, roughly speaking, every diminution of 0.1 per cent. in the oxygen caused a diminution of 3.5 per cent. of the value of the light in pure air. Starting with 20.93 per cent. of oxygen, the percentage of light given off was gradually diminished until with 18 per cent. of oxygen the light was extinguished. A miner's oil safety lamp was used.

Some experiments made in the author's laboratory show atmospheres that extinguish flames of candles and carbide lamps. The candle and carbide lamp were placed in a chamber and when the oxygen therein had fallen to a point where the flames became extinguished the residual air was analyzed.

[7] *Colliery Guardian*, Oct. 25, 1912.

Flame	Analyses of Residual Air		
	CO_2	O_2	N_2
Candle....	2.95	16.24	80 81
Acetylene..............................	6.30	11.70	82.00

The carbon dioxide present was due to combustion and had little effect itself in extinguishing the flame. Other experiments performed in this laboratory showed that a very large percentage of carbon dioxide (replacing nitrogen) is required to extinguish flame; 5 per cent. of carbon dioxide raised the percentage of oxygen in the air in which a flame would be extinguished (using an oil-fed flame) from 16.3 to 16.9 per cent.; with 10 per cent. of carbon dioxide to 17.3 per cent. oxygen; with 21 per cent. of oxygen 43 per cent. of carbon dioxide was required to extinguish the flame. In other words, it is the absence of oxygen in the air of the mines in Cripple Creek that causes flame to be extinguished; carbon dioxide is not present in sufficient quantity to exert an appreciable effect.

Atmospheres in which candles will not burn contain less than about 17 per cent. of oxygen, and atmospheres in which carbide lamps will not burn contain less than about 12 or 13 per cent. of oxygen.

Comparison Between the Indications Afforded by Candle and Acetylene Flames and Analyses of the Samples

At each place where samples were collected the senior author made a note of the condition of his lamp flames, both candle and acetylene. It is interesting to compare these observations with the analyses of the various samples.

Table VIII.—*Observations on Condition of Lamp Flame in Atmospheres of Different Oxygen Content*

Sample Number	Oxygen in Sample, Per Cent.	Lamp Flames
		Midget Mine
781	16 57	Candle would not and carbide would burn.
776	18 92	Candle and carbide burned.
770	10.86	Candle and carbide would not burn.
296	18 53	Candle and carbide burned; candle burned feebly.
400	10.86	Candle and carbide would not burn.
794	11.63	Candle and carbide would not burn.
664	15.06	Candle would not burn but carbide would

TABLE VIII.—*Observations on Condition of Lamp Flame in Atmospheres of Different Oxygen Content.*—(*Continued*)

Sample Number	Oxygen in Sample, Per Cent.	Lamp Flames
		Anaconda Mine
966	7.50	Candle and carbide would not burn.
937	11.78	Candle and carbide would not burn.
691	5 51	Candle and carbide would not burn.
750	7.19	Candle and carbide would not burn.
694	18.30	Candle burned dimly.
948	18.44	Candle burned dimly.
690	17.70	Candle flame just at point where slight movement would extinguish it. Carbide flame burned.
		Mary McKinney Mine
761	10.70	Candle and carbide flame would not burn.
762	8.01	Candle and carbide flame would not burn.
780	13 87	Candle would not burn but carbide lamp would.
976	10.71	Candle and carbide flame would not burn.
		Cresson Mine
797	7.03	Candle and carbide lamp would not burn.
795	16.23	Candle would not burn but carbide lamp would.
760	20.69	Candle and carbide would burn.
759	17.19	Candle and carbide burned; candle burned dimly.
754	14 05	Candles would not burn but carbide would.
753	16 74	Candles would not burn but carbide would.
772	16 65	Candles would not burn but carbide would.
944	17.78	Candle and carbide burned; candle burned dimly.

In summarizing these results one finds that the candle flame became extinguished when the oxygen in the atmosphere fell to between 17 and 18 per cent., and the carbide flames when the oxygen fell to between about 12 and 14 per cent.

COMBUSTIBLE GAS IN THE ROCK GAS

Eight of the samples examined by the senior author contained traces of combustible gas. The largest amount was 0.03 per cent. Presumably all of them contained small proportions that could not be detected by analysis. The relation between the contraction and carbon dioxide in the analyses indicated that this combustible gas is methane. The author was informed that very rarely a small outburst of gas was encountered that burned when a torch was applied to it, but very quickly burned out. Methane (CH_4) is the combustible constituent of the fire-

damp of coal mines. It is explosive in the proportions 5.50 per cent. low limit and about 14 per cent. high limit, *i.e.*, when less than 5.50 per cent. or more than about 14 per cent. of methane is present in air, flame does not travel through the mixture when this is ignited at one point.

Temperature and Humidity in Cripple Creek Mines

Conditions of temperature and humidity were exceptionally good in those mines where gas samples were collected. This is not true of many metal mines. When the wet-bulb temperature exceeds 75° F. the work a man can do begins to fall off; at 85° F., wet bulb, hard work is impossible except for short intervals of time. None of the temperatures recorded by the authors were high.

Summary

The escape of gas from the rock strata into the mines of the Cripple Creek mining district is a menace to life and to mining. The outflow of this gas, supposed by Lindgren and Ransome of the U. S. Geological Survey to represent the last exhalations of the extinct Cripple Creek volcano, is largely influenced by outside atmospheric pressure, because it is confined in the rocks under very low pressure.

At a few mines a pressure system of ventilation has been installed to assist the ordinary natural circulation. Under this system air is blown into the mines by a fan placed at the top of the shaft, thereby placing the mine workings under a little pressure, at some places equivalent to 6 or 7 in. of water (about 0.5 in. of mercury). This method assists very materially in keeping the gas forced back in the rocks.

All told, the 32 samples of mine gas were collected by the senior author from four mines.

It was impossible to secure samples of pure rock-strata gas as it issued from the rocks during the author's inspection of the mines, hence entrance was made as far as it was possible to penetrate into some of those drifts that were most contaminated with the gas. A sample containing 2.69 per cent. of oxygen (sample 760, Cresson mine) was obtained and was the sample containing the largest percentage of the rock-strata gas. Undoubtedly, if one had been able to penetrate 15 or 20 ft. farther into the drift, a sample practically devoid of oxygen would have been secured.

A number of samples that contained the smallest percentages of oxygen were recalculated air-free, in order to determine the actual composition of the gas as it issued from the rocks. This calculating showed that the gas contained between 12.03 and 18.37 per cent. of carbon di-

oxide and 81.63 and 87.97 per cent. of nitrogen. The average of all the results was 13.87 per cent. of carbon dioxide and 86.13 per cent. of nitrogen, or, in round numbers, 14 per cent. of carbon dioxide and 86 per cent. of nitrogen.

The gas, then, as it occurs in the rocks is a mixture of carbon dioxide and nitrogen with the latter much in excess.

The bad effects on life and lights are principally due to the fact that the rock gas so dilutes the air of the mines that the oxygen therein falls to a point where lights will not burn or so low that life is endangered.

Symptoms of distress in men do not usually begin to appear until 3 to 4 per cent. of carbon dioxide is present, when the breathing becomes affected. Men can go on working for a considerable time in such an atmosphere although they will certainly become more quickly fatigued. Carbon dioxide affects the so-called respiratory center in the brain and makes a man breathe a larger volume of air over a given time than if no carbon dioxide were present. While air with so small an amount of carbon dioxide as 2 per cent. may be breathed with comparative safety and no great discomfort, at the same time the efficiency of the workmen is lowered. In addition to the work a man may be doing in such an atmosphere he is handicapped to this extent: He is forced to breathe a larger volume of air over a given time, a feat that consumes energy just as his work of drilling a hole in the rock or loading ore consumes energy. In England the Coal Mines Act requires that the carbon dioxide shall not rise above 1.25 per cent. in any part of a mine.

When the oxygen in air is gradually reduced very little effect may be noticed before the occurrence of impairment of the senses and loss of power over the limbs. If the reduction is gradual and the symptoms carefully watched it will be noticed that at about 12 per cent. of oxygen (that is, with a reduction of about 9 per cent. from the composition of atmospheric air) the respirations become just perceptibly deeper and more frequent, and the lips slightly bluish. Distress increases with continued decrease of the oxygen until with 6 or 7 per cent. there is marked clouding of the senses and loss of power over the limbs, which would end sooner or later in death. In a test similar to the above that the senior author participated in, a man lost consciousness when the oxygen dropped to about 7 per cent.

As the oxygen in air decreases the illuminating power of lights diminishes, until in the case of the ordinary oil-fed lamp wick such as a candle or miner's oil torch or safety lamp, the flame becomes extinguished at about 17 per cent. of oxygen, and the acetylene flame at between 12 and 13 per cent. of oxygen, *i.e.*, if a candle flame goes out the oxygen content of the air is less than 17 per cent., and if the acetylene flame goes out the oxygen content is less than between 12 and 13 per cent. This effect is almost entirely due to the oxygen content and even in the

Cripple Creek mines where the carbon dioxide may be from 5 to 10 per cent., in atmospheres that extinguish flames, it exerts only a minor effect.

Men should not work where candles will not burn, and should not be solely reliant upon the acetylene flame for guidance regarding bad air. The acetylene flame will burn in air where the oxygen is only a few per cent. above the proportion that is very dangerous to life, hence only a small margin of safety is assured.

Eight of the samples contained traces (0.01 to 0.03 per cent.) of combustible gas, apparently methane (CH_4). The proportions were so small, however, as to be insignificant.

Conditions of temperature and humidity were good in those mines where samples were collected.

The authors are indebted to several mining men in Cripple Creek for generous assistance in collecting data, especially to Messrs. Suydam, Allen and Arthur.

Discussion

J. S. Haldane, M. D., F. R. S., Oxford, England (communication to the Secretary*).—The valuable paper of Messrs. Burrell and Gauger is of special interest to me as, through the courtesy of the management, I had an opportunity of visiting the Portland mine, Cripple Creek, in 1911, in connection with the investigations on acclimatization to high altitudes, carried out on Pike's Peak, near Cripple Creek, by Professors Yandell, Henderson and Schneider, Dr. Douglas and myself.[1] The analyses of one or two samples of the vitiated air were entirely confirmatory of those of the authors. A fatal accident owing to two men being lowered down a shaft filled with the gas had occurred in a neighboring mine just before our visit.

The composition of the gas is so similar to that of the "black damp" met with in coal mines, metalliferous mines of different kinds, and wells, that there can be little doubt that its origin is similarly due to oxidation processes in the strata. It is very remarkable, however, that the gas is produced in such large amount in the Cripple Creek mines. The rock is apparently very porous or full of fissures, and also contains some substance which oxidizes freely. In some metalliferous mines, and to a less extent in coal mines, this substance is pyrites, the CO_2 being due to subsequent liberation of this gas from carbonates by the sulphuric acid formed; but in many cases the oxidation process is certainly a different one, and further examination would be needed in order to determine

* Received Aug. 25, 1916.

[1] Philosophical Transactions of the Royal Society, Ser. B., vol. 203, pp. 185–138 (1913).

what oxidation process occurs in the rock at Cripple Creek. It follows from the authors' analyses that the black damp formed contains sometimes so little CO_2 (*e.g.*, samples 753, 754, 761, 772, 794) that it is lighter than air, as is not infrequently the case with black damp in coal mines, wells, etc.

Further observations on the relation of the issue of the gas to changes in barometric pressure would be of much interest. As I showed in a paper contributed to the *Transactions of the Institution of Mining Engineers* (Great Britain) in 1896, the special danger to well-sinkers from black damp arises from the fact that although they are well aware of the need for testing the air with a candle before descending, they do not realize that the test must be repeated at each descent. A well which, for instance, has been quite clear of gas in the morning may, in consequence of even a slight fall of barometric pressure, be overflowing with black damp in the afternoon. The enormous quantities of gas or air which may issue from a well with a fall of barometric pressure are very surprising.

It is a question of some interest whether it is better to apply ventilation by pressure or by exhaust in cases like that of the Cripple Creek mines. The objection to pressure is that in the event of the fan being temporarily stopped there may be an immediate discharge of black damp owing to the fall in the air pressure. In practice, however, there may be counterbalancing advantages.

As regards the effects of the black damp on the men, it must be borne in mind that the miners will all be acclimatized to the oxygen deficiency due to the altitude (about 10,000 ft.) of the Cripple Creek district. The results of the Pike's Peak expedition showed that there are three factors in this acclimatization:

1. The breathing is increased, so that at Cripple Creek, the lungs are 30 per cent. better ventilated than at sea level.

2. The layer of living cells which separate the blood from the air in the lungs plays an active part in forcing oxygen inward into the blood, where at sea level this layer is only passive, so that the oxygen passes in by simple diffusion.

3. The blood is about 20 per cent. richer in hæmoglobin and red corpuscles at Cripple Creek. Taking into account the influence of acclimatization, the effects of a given percentage of black damp in the air will probably be nearly the same at Cripple Creek as at sea level.

In a paper contributed this summer to the *Transactions of the* (British) *Institution of Mining Engineers,* I pointed out that the excessive tendency to emphyrema and bronchitis among older miners is probably due largely to breathing air contaminated by too much black damp. I was mainly responsible for the provision in the British Coal Mines Act of 1911 that the percentage of CO_2 should not be allowed to exceed 1¼ per cent.

This maximum standard is easily attainable, and seems well worth maintaining. A very simple method of ascertaining how much black damp is present in the air is afforded by the "tube and taper" method which I introduced about 5 years ago.[2] This method was very useful to me in searching for a sample of vitiated air at the Portland mine, where it was by no means easy to find such a sample.

The general health conditions at the Cripple Creek mines appeared to be much better than at most other metalliferous mines known to me. The absence of miners' phthisis was a very marked and extremely interesting feature in this district, and suggested new ideas as to the causation of this very formidable disease.

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[2] Fully described in my book *Methods of Air Analysis*, London, 1912.

The Solution of Some Hydraulic Mining Problems on Ruby Creek, British Columbia

BY CHESTER F. LEE* AND T. M. DAULTON,† SEATTLE, WASH.

(Arizona Meeting, September, 1916)

Introduction

THE Atlin Mining District is in the northwest corner of the Province of British Columbia. Ruby Creek, where the operation to be described is situated, is 17 miles east of the town of Atlin and about 18 miles south of the south line of Yukon Territory (Fig. 1).

Gold was discovered in the district in January, 1898, and the production to date from placer mines has been $4,518,000, according to the reports of the Minister of Mines for the province.

Geology

The essential points in the regional geology are given in the following paragraphs. Long erosion produced a plane-like surface slightly above sea level with a few peaks and ridges.[1] A gradual uplift followed and erosion increased, the streams cut deeper, the cuts so made being afterward increased by glaciation. The ice straightened and planed the slopes of the main valleys and widened and lowered their floors, in some cases flooring the valleys with silt, sand, gravel, and clay. As the glaciers rapidly retreated, lakes were formed.

As to the origin of the gold, J. C. Gwillim says:[2] "The Surprise granite has been carried northward toward Gladys Lake and Teslin, and the granite of the great plateau on the Taku trail was found in blocks on the opposite range across Hurricane River 10 miles north of its original position. Otherwise there is little evidence of far removed material, the boulders of the lake shore and creek beds being of local appearance. This appears to strengthen the opinion that the placer gold had its origin within the drainage basins where it is now found."

In the gold-bearing creeks of the Atlin district there are two "runs"

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* Mining Engineer.

† General Manager, Placer Gold Mines Co.

[1] D. D. Cairns: *Memoir No. 37, Geological Survey of Canada,* pp. 19–20 (1913).

[2] *Annual Report of Geological Survey of Canada* (New Series), vol. 12 (1899), p. 73a.

of gold.[3] The first is in a yellow gravel, the color of which is due to iron. This gold has a reddish tinge; it shows on Spruce and Pine Creeks as well as on Ruby Creek. The flow seems to have proceeded down the two former creeks and westerly and was followed by a period of heavy deposits of clay and barren gravel.

The second run, in bluish gravel, seems to have come from the

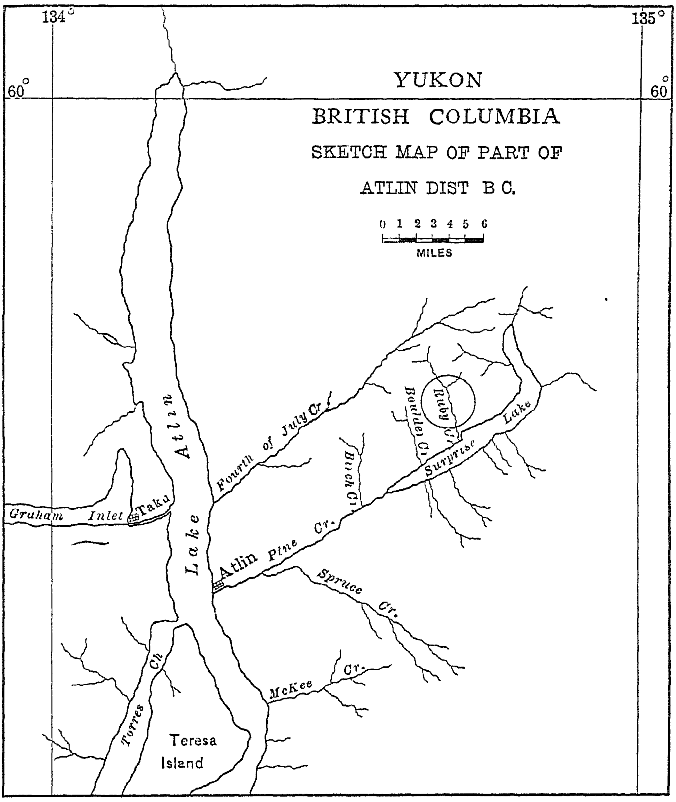


FIG. 1.

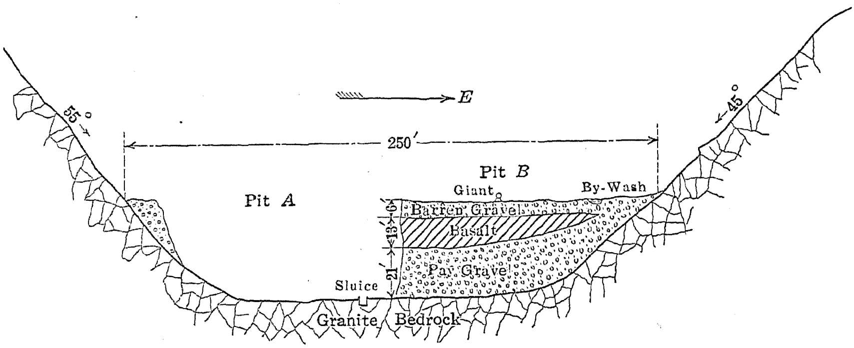
direction of Boulder and Birch Creeks and extended to McKee Creek. This gold is bright, but is not as high grade as the other.

After the second run there was a subsidence, then a lowering of the water level or a raising of the land with a wearing down of the present creek channels which in places cut through the previous gold runs and concentrated the gold in the present channels.

Ruby Creek is 8 miles long and flows in a southerly direction into Surprise Lake. The gravel being mined is about a mile upstream from

[3] W. F. Robertson: *Annual Report of Minister of Mines of British Columbia,* 1900, pp. 754 to 755.

the lake and is the stream bed or "creek gravel." The rock underlying the gravels is granitic and has taken its present form through glacial action, the alluvial material having been deposited subsequently, partly



CROSS SECTION

FIG. 2.—CROSS-SECTION OF RUBY CREEK SHOWING POSITION OF GRAVELS BEING WORKED BY HYDRAULICKING.

by glacial and partly by stream action, in successive flows and at widely separated times, as shown by a dike of basalt about 13 ft. thick which overlies the bedrock gravel on the east side of the creek.

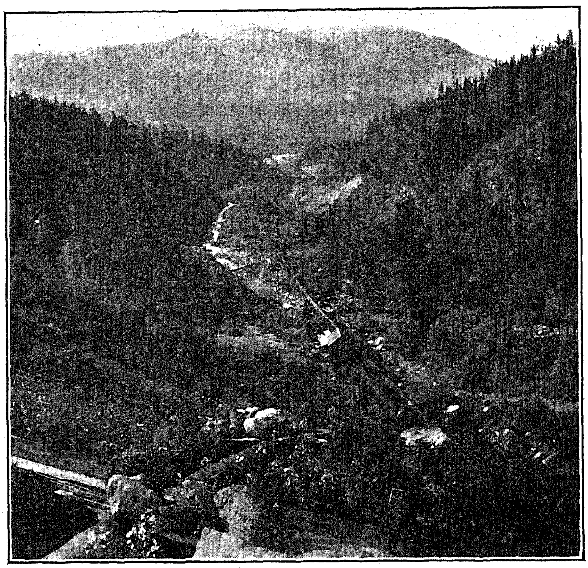


FIG 3.—LOOKING DOWN RUBY CREEK FROM PRESSURE BOX.

The gulch has steep banks and is about 250 ft. wide from rim to rim at the surface of the gravel. The depth of gravel at the center is 42 ft. (Figs. 2 and 3).

Water

The log crib storage dam is situated 4 miles from the pit up Ruby Creek. It is 150 ft. long, 12 ft. high and 6 ft. wide at top, producing a reservoir ¾ mile long and ½ mile wide with an average depth of 8 ft. From this the water follows the creek bed 3½ miles to the intake whence it flows into the ditch which is 8 ft. wide at the top, 4 ft. wide at the bottom and 3½ ft. deep, the grade being 8 ft. to the mile. There are 400 ft. of ditch, then 300 ft. of flume across the gulch and 800 ft. of ditch to the pressure box, which is 8 by 10 by 10 ft. From this box issues the pipe line, 2,265 ft. long, beginning with 26-in. diameter 16-gage pipe and ending with 16-in. 14-gage pipe. The normal flow is about 1,150 miner's inches (31 cu. ft. per second), which is used through 7-in. nozzles of No. 6 Hendy giants. Also, 1,000 in. comes down the creek and is used as a bywash to assist in handling the heavy material. The vertical head at the pit is 250 ft. Beside the main pipe line a 16-in. line tapering to 6 in. in a length of 432 ft. is taken out of the pressure box and used on a Cassel impulse wheel of 25 in. diameter to actuate a Sullivan 8 by 8 class WG-3 belt-driven compressor, the air from which is used to drill boulders.

The sluices from the working pits are 42 in. wide by 42 in. high set on a grade of 4 in. to 12 ft. (2.77 per cent.). The sluices are double, as will be explained, and their length is 4,000 ft.

Difficulties Overcome

The obstacles to be overcome in developing and operating this property and the way they were surmounted make this operation noteworthy.

The conditions which presented difficulties are outlined below:

1. The great width of the deposit, 250 ft., which with the 42-ft. banks made impossible the working as a single pit from rim to rim.

2. In May and June the flood waters produce four times as much water as can be used and the excess water, if carried on top of the bank, tends to undermine it and cave it into the pit.

3. Only a small grade (about 3 per cent.) was available, which offered poor facilities for dumping.

4. The boulders were large in number and size.

These problems were met as follows:

1. The ground was divided by a median line up the creek into two series of pits, A on the west side and B on the east, each being about 125 ft. wide. Pit A was advanced 400 ft. first and then pit B was begun, the pits being worked alternately. After hydraulicking some hours in pit A until the pit is so full of boulders that the stream is no longer effective, the water is turned off at the valve above the workings and two men are left there to block-hole and blast the boulders. Hydraulick-

ing is then begun in pit B and continued until the boulders obstruct the work, when they are drilled and blasted in turn and so on.

2. The arrangement of sluices is such that in flood season the excess water is allowed to flow over the face of pit A into the sluice at the bulkhead and down this to where the double sluices begin opposite pit B (Figs. 4 and 5). The gates at the head of the east sluice are closed and the flood waters go through the west sluice to waste. During the time of excess water, work in pit A is suspended, but pit B can be worked without interruption, save for the time needed to drill and shoot the boulders. Without these arrangements work would have to be sus-

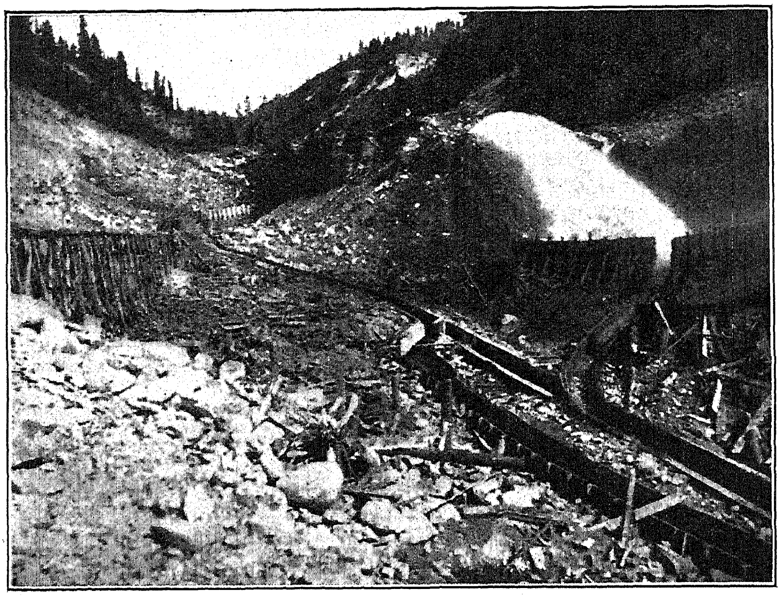


FIG. 4.—HYDRAULICKING OPERATIONS OF THE PLACER GOLD MINES CO. ON RUBY CREEK. (Pit A on left; Pit B on right.)

pended during high water, which would be a serious drawback as the season is only about 150 days in all. When the water recedes to the point that it can all be handled through the giant and bywash, hydraulicking is resumed in the pits alternately as above described. Only the east sluice is used for handling gravel and is fitted with gold-saving devices. The arrangement of gates, as shown in Fig. 5, permits the gravel from either pit to be sent down the east sluice; the west sluice is used only for excess water. The upper gate, shown in detail in Fig. 5, rests loosely against the post and is raised off the bottom of the sluice by two men throwing their weight on the end of the long lever. This relieves the water pressure and enables the gate to be thrown over. If it were not for this arrangement the pressure of the stream would make it impossible to move the gate.

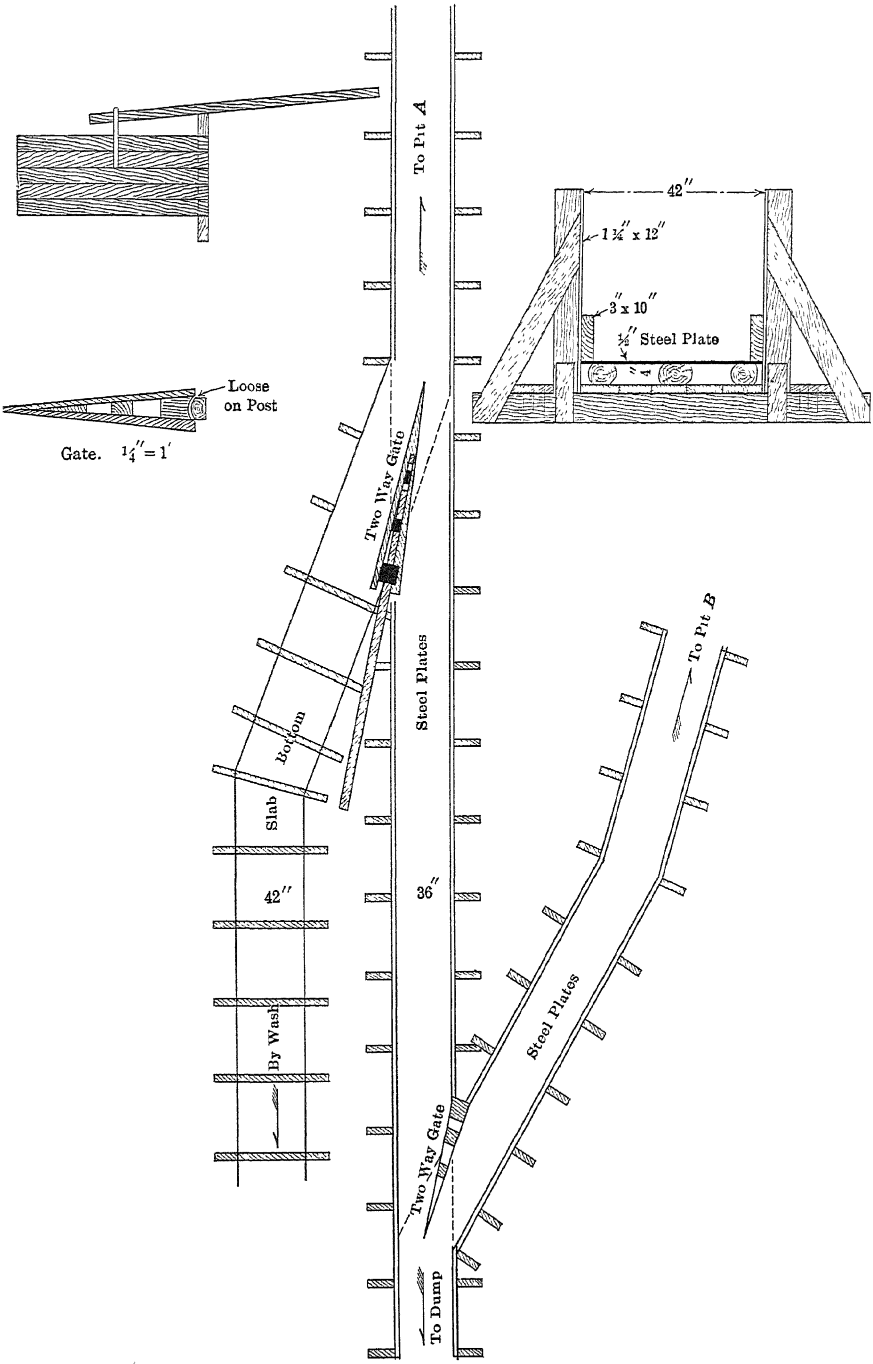


FIG. 5.—ARRANGEMENT OF SLUICES; DETAILS OF TWO-WAY GATES; AND CROSS-SECTION OF GOLD-SAVING SLUICE.

3. The gold-saving sluice boxes, 42 in. wide, were originally paved with spruce blocks 8 by 8 in. by 10 in. long, spaced 2 in. apart longitudinally, but the excessive wear on them entailed frequent stoppages for renewals which was both annoying and expensive as the season is short and it is imperative to get the maximum use of the water during the open period. The steepest grade obtainable was 3½ per cent., which seriously limited the amount of gravel handled. In 1914, therefore, 2,400 lin. ft. of high-carbon steel plates (0.9 per cent. carbon) were bought from the Carbon Steel Co. of Pittsburgh and substituted for wood blocks for this distance. The grade of the flume was changed to 2.77 per cent., which gave 20 ft. additional dump at the lower end of the sluice.

The plates cost $45 per ton at the mills, $108.80 per ton laid down at Ruby Creek. Plates of this sort were first used by The McKee Creek Mining Co. in this district in 1909. The plates are 12 ft. long, 38 in. wide and ½ in. thick and are placed 2 in. apart as to their ends with a drop of ½ in. from one plate to the next. Fig. 5 shows how they are supported and held in position; 6- to 8-in. logs are sawed flat on two sides to a thickness of 4 in. and made ½ in. thicker on the downstream end than on the upstream in each 12 ft. of length. The plates are supported by these and held down by the edges of the 3- by 10-in. lining boards.

The space between plates makes an excellent riffle. The use of the plates increases the capacity of the sluice about 40 per cent. and enables angular pieces of blasted boulders 30 in. in their longest dimension to be put through as against 20-in. pieces with block riffles. Occasionally extra large boulders get into the sluice, and 5- by 2½-ft. ones have gone through without trouble. All trouble from jammed sluices and overflows has thus been obviated.

After a season's wear and carrying 67,940 cu. yd. of gravel, the plates showed an abrasion of 1/32 in. At the end of the 1915 season, after transporting a total of 130,380 cu. yd., holes developed at some points. The surface skin of the plates is harder than the interior and where the surface becomes slightly worn deterioration is more rapid. A steel equally hard throughout is desirable for this use and the question of its production has been taken up with the manufacturers.

4. In 1915, 6,380 boulders were drilled and blasted and 21,955 "bulldozed" without drilling; in 1914, 23,832 were blasted, which, taking the two years together, is a boulder for each 2.5 yd. of gravel worked. The practice is to "bulldoze" the flatter and smaller boulders without drilling, and block-hole the larger ones and pipe all the pieces through the sluice (Fig. 6). For drilling, two Sullivan DA-19 40-lb. hammer drills are used with air pressure at 90 lb. at the compressor. The air line is 2 in. in diameter and 1,000 ft. long; two lines of 50-ft. hose ½ in. in diameter connect directly with the drills. In 1915, explosives cost 26c. per boulder.

In part of the ground an additional obstacle must be overcome. On the east side in pit B a dark basaltic dike about 13 ft. thick lies on top of the 21 ft. of pay gravel and is itself overlaid by 6 ft. of waste gravel (Fig. 2). This basaltic flow is lenticular and thins out both in the up-

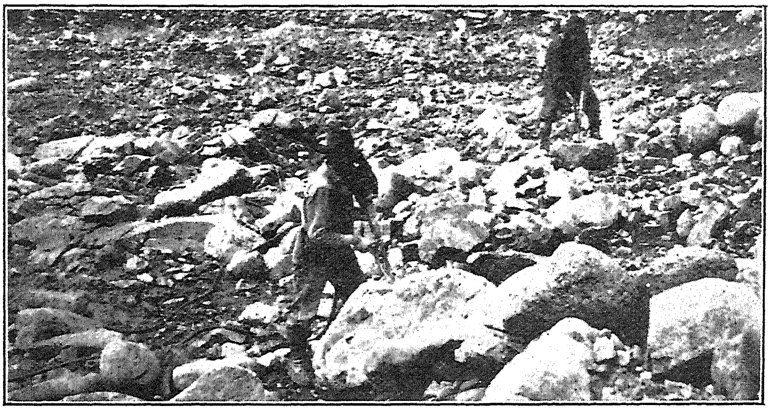


FIG. 6.—BLOCK-HOLING BOULDERS IN THE PIT AFTER HYDRAULICKING.

stream direction and from the center toward the east rim. It pinches out entirely 250 ft. upstream. Fortunately the basalt is friable and fairly soft, so that by putting gopher holes under it and shaking up with powder it is gradually broken through, and can be washed away by ground sluicing and hydraulicking.

In 1915 the following results were obtained:

Average number of men employed	20
Hours piping in pay gravel	978
24 hr.-in. of water on pay gravel	46,862
Total cubic yards of gravel handled	62,440
Cubic yards per 24 hr.-in.	1.33
Cost per cubic yard of gravel handled:	
Labor	$0.302
Explosives	0.119
Lumber	0.007
Stable	0.009
Hardware	0.006
Licenses and rentals	0.011
Liability insurance	0.005
General expense	0.017
Total	$0.476

The property belongs to the Placer Gold Mines Co. of Seattle; G. W. Fischer, President, T. M. Daulton, General Manager. The latter has planned the work and conducted the operations since the company took over the property from the original locators in 1908, and is still in charge.

The Rifling of Diamond-Drill Cores

BY WALTER R. CRANE,* PH. D., STATE COLLEGE, PA.

(Arizona Meeting, September, 1915)

OPERATORS of diamond drills have long been familiar with thread-like markings or riflings on cores but apparently have given but little serious thought to the conditions that are responsible for their production. The opinion generally held by those who have observed the phenomena is that the riflings are produced directly or indirectly by vibration, but so far as the writer is aware no systematic attempt has been made to demonstrate the fact.

A careful search through the literature on diamond-drilling practice has been rewarded by only one reference to such markings of cores. J. N. Justice[1] writes, "Of the peculiarities met with in the core, I will mention only one. The core from the "H" drill, at about 360 ft. came out rifled and pentagonal. The change from the circular to the pentagonal was sudden, and for 9 ft. this structure was maintained, when suddenly it returned to the circular. I have not been able to find any one who can explain the cause of this change of form. The drill men marveled at it and suggested vibration, six-stone bits, and several other hypotheses which need not be mentioned."

The writer became interested in the matter of core riflings some 10 years ago while in the Flat River lead region of Missouri, where many remarkable examples of "rifled" cores were observed and collected. Since that time similar phenomena have been observed in various localities throughout the United States and Canada, and samples have been gotten together illustrating a large number of variations in form.

After a careful study had been made of the material at hand it seemed desirable that certain data should be obtained from the experience of others regarding the normal action of diamond drills in forming cores, portions of which are occasionally rifled, before an attempt was made to draw any conclusions as to their cause.

About 20 letters were addressed to diamond-drill companies, operators, and engineers actively engaged in drilling operations, with a

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* Dean of the School of Mines, The Pennsylvania State College.

[1] J. N. Justice: Diamond Drilling in West Africa, *Transactions of the Institution of Mining and Metallurgy*, vol. 12, p. 309 (1902–3).

request for a statement of their opinion concerning the cause of systematic markings or rifling of cores. The concensus of opinion obtained from the replies received was that vibration is responsible for the phenomena. In a general way vibration seems to be the only logical explanation, yet owing to the fact that in the operation of drilling, spiral markings on cores are the exception rather than the rule (although in certain formations they are of frequent occurrence), it has not been easy to determine the exact relation existing between vibration and the markings.

The character of phenomena observed on examination of a large number of rifled cores may be summarized under their respective heads in the following paragraphs.

Number of Riflings to the Turn

While the markings are commonly spoken of as riflings, it might be better to designate them as threads, owing to their association with other terms. The number of threads to the turn varies between wide limits; the smallest number observed was three (No. 3, Fig. 1), the largest 13—the average being seven. Five threads to the turn appear to be the most common, although that may be a mere coincidence as a result of examining a comparatively large number of cores from one district, where similar formations were drilled through, and under like conditions of drilling, particularly with respect to equipment. With large pitches the five-threaded riflings may assume a pentagonal section and are commonly spoken of as "pentagonal" cores.

While the number of threads was observed to be uneven in every core examined, yet it has been demonstrated that there may be even numbers as well. In fact, there may be both odd and even numbers of threads on one and the same core, although on the samples examined change in number was brought about by threads bifurcating or coalescing, thus maintaining the relation as to number. Further, it seems to be the rule that prominence of threads is coincident with a diminution in number (Nos. 3 and 8, Fig. 1).

Pitch of Threads

The pitch of threads noted on a large number of cores varies even more than does the number of threads to the turn; the range observed was between $\frac{7}{16}$ in. (for nine-threaded core) and $3\frac{5}{8}$ in. (for five-threaded core). A much wider range is possible, and, in fact, no limit can be set. The pitch of threads varies directly, of course, with the number of threads to the unit length or inch. The smooth or unrifled core may be considered as the lower limit (lower end of No. 5, Fig. 1), the threads blending one into the other; the strongly rifled form with pitch approaching infinity is the upper limit, both limits being common but not so universal as the

intermediate forms (see Nos. 1, 3, and 8, Fig. 1). Further, there is a definite relation between the number of threads to the turn and the pitch.

Depth and Shape of Threads

The depth and shape of threads are concordant characteristics of rifled cores and while they may be more difficult to explain than some of the other features, they are even more clearly defined. The depths of threads as measured vary between $1/100$ and $1/16$ in. apparently varying

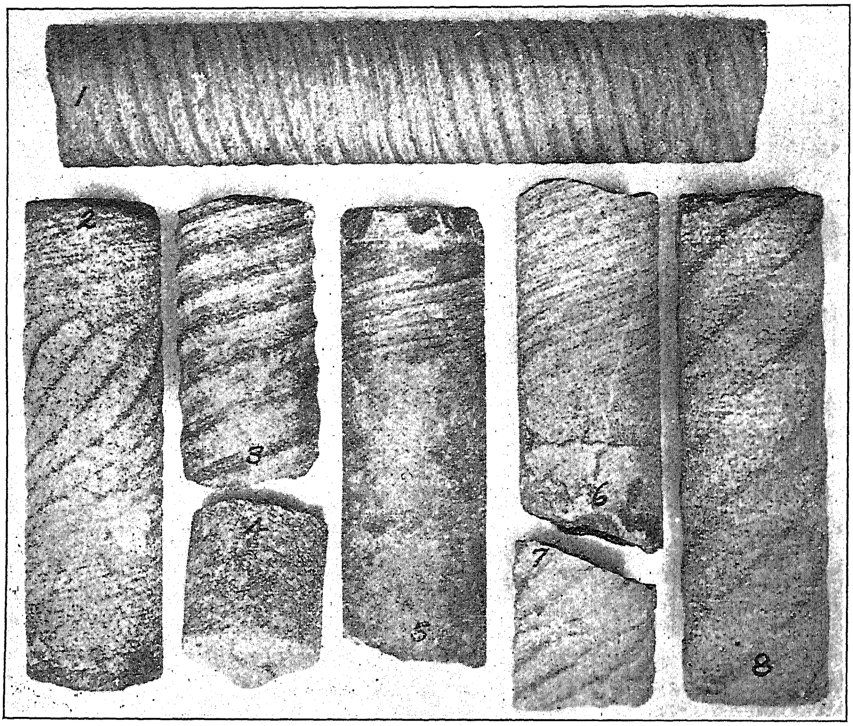


FIG. 1.—SAMPLES OF CORES IN LIMESTONE SHOWING RIFLINGS.

to a limited degree with the number of threads to the turn, as mentioned above.

The shape may be of a variety of forms, the rounded form predominating in the coarser threads (No. 3, Fig. 1), while roughly V-shaped threads occur when the riflings and pitch are small (see ends of No. 2, Fig. 1). A braided appearance is occasionally observed, due to a set of threads of a certain pitch being superimposed upon another of different pitch (see No. 6, Fig. 1 and Fig. 2). It is not of infrequent occurrence that double and triple threads are observed which in all respects are similar to the ordinary forms; however, with such combinations the troughs

between the riflings are usually wide and shallow (see Fig. 3). Flat-topped or square threads are also formed.

Various reasons and theories have been advanced to explain the formation of threads on diamond-drill cores, the following being the most commonly held: (1) vibration of drill rods; (2) four- and six-stone bits; (3) the up-and-down action of water currents carrying cuttings; (4) particles of mineral or metal working their way down through the core barrel; (5) differential action between core and core barrel, the former having been broken off and turned through friction with revolving bit and core barrel; and (6) overset of stones in bit.

After careful investigation and consideration of the data at hand, certain of the causes outlined above may be eliminated as inadequate, thus narrowing them down to one or more possible or probable ones.

It is but natural to expect that machines operating under high

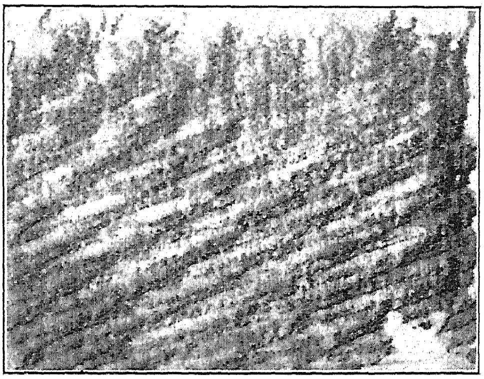


FIG. 2.—BRAIDED MARKINGS ON CORE DUE TO TWO SETS OF THREADS CROSSING ONE ANOTHER. (RUB OF CORE.)

pressure, as diamond drills do, will set up considerable vibration, particularly when boring through difficult formations. So intense is the vibration of the drill and rods at times that the whole outfit and even the ground for many feet around will be strongly shaken. Vibration is therefore a well-known phenomenon attendant upon diamond drilling and its presence does not have to be assumed.

Water currents laden with cuttings do not, except very rarely, as when reversed, enter the core barrel or in fact the inside of the bit, as the wash water passes from the inside to the outside of the bit, and thence to the surface between the drill rod and the hole or casing. Therefore, during the operation of drilling only practically clear water comes into contact with the core.

The theory advanced that the number of stones (carbons) in the bit is responsible for the riflings on cores can hardly be considered as sound for the reason that bits with four, six, or eight stones (considering the

inside stones only) seldom if ever produce a number of threads corresponding to the number of stones; in fact, in all cases observed the threads were odd numbers. Further, as the stones are evenly spaced about the bit, it is obvious that the number of threads would be determined by the number of stones and that the number could not change either by an increase or decrease, odd or even. It would probably be more logical to assume that there is but one cutting point instead of a number.

What the effect of small particles of mineral detached from the core and pieces of metal broken from threaded connections of rods, teeth of clutches at the surface, etc., may be when they enter the rod and are carried to the core barrel and bit by wash water, is largely conjectural except in those cases where cores have been jambed in bits and core

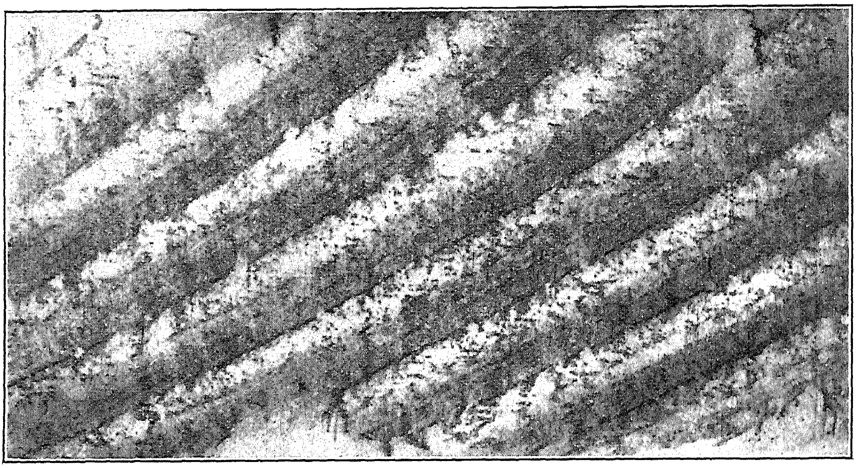


FIG. 3.—RIFLINGS SHOWING TRIPLE THREADS AND LACK OF PARALLELISM DUE TO CHANGING PITCH. (RUB OF CORE.)

lifters by such fragments, or where the bit or core barrel has been severed by the cutting action of a particle lodged in the core. As pointed out above, it might be more reasonable to assume that one cutting point is responsible for the riflings, which condition could be brought about by a particle of hard mineral or a fragment of steel lodging in the core lifter or bit and acting as the cutting medium.

The overset of stones in a bit is so slight that it would hardly seem possible for threads to be cut to the depths observed, unless it is assumed that through accident a stone has become loosened and the overset greatly increased. The usual overset of stones in bits is $\frac{1}{64}$ to $\frac{1}{32}$ in., while the depth of threads may reach and even exceed $\frac{1}{16}$ in. It is difficult to imagine any possible movement of bit that would allow a stone normally set to cut to that depth. It is reasonable then to assume that

particles of mineral, fragments of metal or loosened diamonds may be the media by which the threads are cut.

The breaking off of the core and its rotation within the core barrel is of common occurrence and can be observed in parts of nearly every section of core lifted, especially in certain loosely bedded or broken formations. That the wear of the loosened portions of core is great is evident on comparing the actual advance of rod with the length of core extracted (see No. 4, Fig. 1). Further, it is a curious coincidence that practically every case of rifled core examined bore evidence of considerable wear on one or both ends, showing that the cores had been free to rotate with the core barrel, but had been retarded to a certain extent by grinding between the over and underlying portions. That differential action should be sufficiently positive and regular to permit of well-defined and symmetrical markings is exceedingly doubtful.

It is evident from the foregoing statements that of the possible causes there are only two that have sufficient weight and prominence to be considered worthy of further consideration, namely: vibration and the cutting action of particles of mineral or metal lodging in the bit or core barrel, or the excessive overset of loosened stones in the bits. As it is necessary to have a cutting medium whether it be mineral, metal, or overset of diamonds, we can assume that the latter conjecture is more or less tenable. The cutting medium tentatively agreed upon, the formation of the threads resolves itself into the question of how vibration might act to produce such phenomena.

Certain observed facts seem to be opposed to the theory of vibration of drill rod as a probable cause of riflings; however, as the actual conditions existing at the time of the formation of the markings are not definitely known, some of the points mentioned below may have no great weight.

The facts that do not seem to conform with the theory of vibration as a cause of riflings on diamond-drill cores are as follows: (1) rifled cores occur more frequently in soft than hard formation, where the resistance to rotation of the drill rods and the consequent vibration of rods might be expected to be the least; (2) contact with sides of the hole, particularly inclined and crooked holes, would prevent to a large extent the free longitudinal or transverse vibration of the rods; and (3) an apparent persistence in number of threads, whether odd or even, regardless of depth of hole, size of rods, speed of advance, etc.

On the other hand, the facts supporting the theory that vibration is responsible for rifling of cores may be outlined as follows: (1) regularity of threads, the same relation existing between threads of different pitch (see No. 2, Fig. 1, and Fig. 4); (2) the number of threads to the turn is remarkably persistent and when an increase or reduction occurs the change is due to the splitting up or blending of existing threads (see Fig.

5); (3) the distance between threads circumferentially remains constant (see No. 2, Fig. 1 and Fig. 4); (4) the threads show that they are cut by some positively and systematically acting medium moving circumferentially, and not by water currents or a grinding action parallel with the threads; (5) the crossing of threads of different pitch, producing a braided effect (see No. 6, Fig. 1 and Fig. 2); (6) the rifling of cores is of more frequent occurrence in vertical holes or those of slight inclination; and (7) rifling is probably more common and pronounced in holes of considerable depth.

The two last-mentioned observations are largely conjectural, but from information at hand, though meager, the assumptions seem to be corroborated.

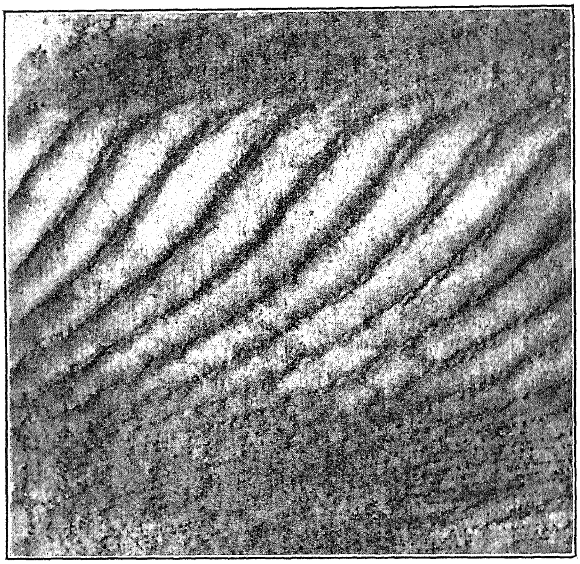


FIG. 4.—WIDE RANGE OF PITCH DUE TO VARIATIONS IN SPEED OF ROTATION. (RUB OF CORE.)

Up to this point the data were obtained from and based directly upon observations taken in the field and a careful examination of cores. While the evidence points very conclusively to the fact that vibration must be the cause of the riflings, yet what form the vibration takes, whether longitudinal or transverse, and the relation it bears to size of rods, rate of advance, revolutions per minute, etc., are wholly unknown. How to secure the information was the next question that had to be answered. It was not until the thought suggested itself that it might be possible to reproduce the riflings, that the solution assumed a tangible form.

A metal lathe was chosen as the most suitable instrument to be

employed for investigating the formation of riflings or threads on cores, although no attempt was made to turn out cores from solid material.

Placing a piece of pipe in the chuck of the lathe with one end free, and the cutting tool set as short as possible to prevent vibration, an attempt was made to remove thin layers from the outside of the pipe with varying speeds of advance and rotation of the pipe. As the tool advanced toward the chuck from the free end of the pipe, vibration became more pronounced and finally very violent, being maintained up to the chuck. On examination of the surface of the pipe after such a cutting

FIG. 5.—CORE SHOWING THE SPLITTING UP AND BLENDING OF THREADS, ALSO SUDDEN CHANGE OF PITCH. (RUB OF CORE.)

had been made, it was found that rough threads had been formed, which closely resembled the riflings observed on the drill cores (see No. 5, Fig. 6).

With this as a beginning an elaborate series of experiments was made, during which a large number of combinations of speed of advance (advance in terms of threads per inch), speed of rotation, size and length of piece in chuck, shape and length of cutting tool, and character of material (both tubular and solid being used), were tried with the result that practically every form of rifling observed on the cores was obtained.

However, with the limitations necessarily placed on the work through size of lathe and test pieces, also speeds, it was not possible to reproduce the full range of threads as to number to the turn, pitch, etc. (see Fig. 6).

To conform still further to the conditions existing in drilling, the end of the test piece not in the chuck was rigidly supported by the tail piece of the lathe, thus holding the piece rigidly and under compression between the two points of support. No difference in results obtained was observed.

It is evident that in this work the conditions existing in the case of the diamond drill were reversed in that the part representing the core was

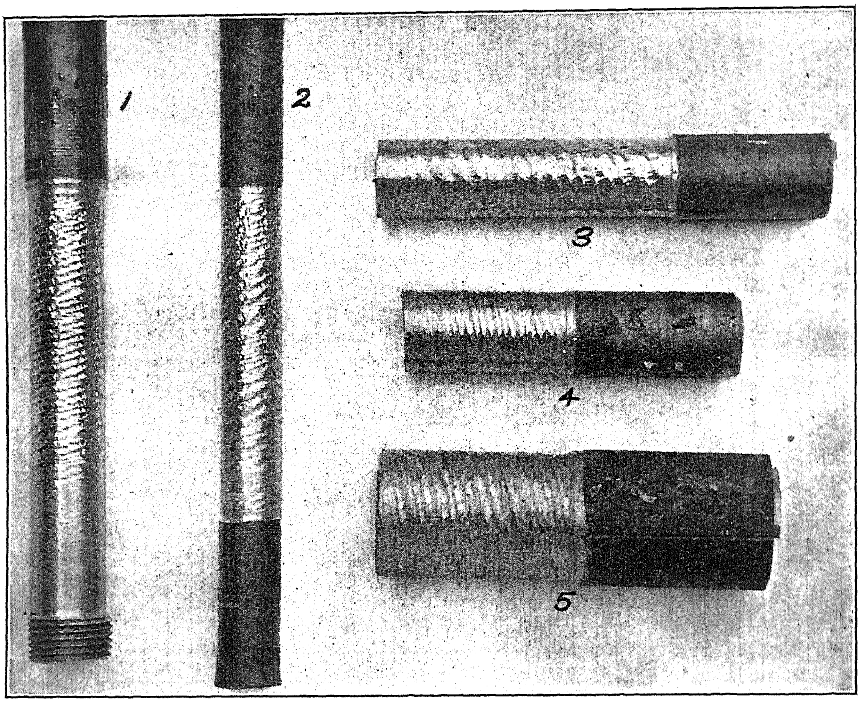


Fig. 6.—Riflings Formed on Metal Tubes and Rods by Vibration.

rotated while the tool remained practically rigid. To verify the results further, a smooth diamond-drill core was substituted in place of the metal tube or rod and a long slender tool was used instead of the rigid one formerly employed. Rough but rather indistinct threads were produced, which although not entirely satisfactory still further confirmed the results previously obtained. Riflings were also produced on the inner surface of cylinders, a long cutter-bar being used, the vibration of which produced a peculiar frosted or wavy marking at slow speeds, but developed into distinct threads at higher speeds and greater vibration.

The action of a lathe when producing riflings or thread-like markings

is commonly spoken of as "chattering" and in lathe work is very objectionable. When change in speed is not possible nor desirable other means are employed to dampen the vibration, a common method being to insert a wooden wedge between chuck and bed plate, which while not preventing rotation stops the vibration.

With the large amount of data obtained from this work it was possible to correlate it with information secured from the rifled cores and thus demonstrate in a conclusive way the cause of the riflings and the relation that exists between the number of threads, pitch, etc.

Conclusions

The phenomena of rifled cores are due to torsional vibration of the drill rods, which in turn is produced by the rotation of the rods. The

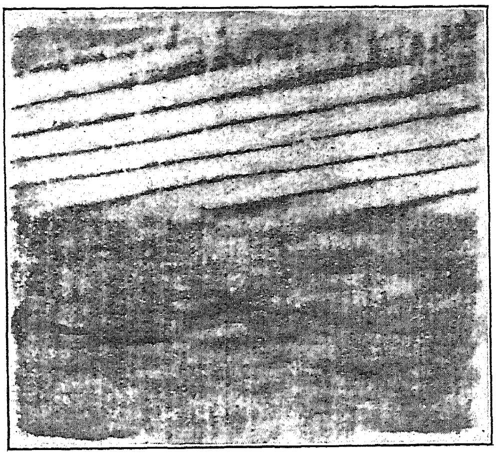


FIG. 7.—ABRUPT CHANGE FROM SMOOTH CORE BELOW TO RIFLED CORE ABOVE. (RUB OF CORE.)

cutting medium attached to the rods engages with the core, penetrating to a certain depth, and thereby temporarily checking the rotation of the rods. When the energy stored up in the rods by the torsional strain exceeds the frictional resistance between the cutting medium and the core, it forces the cutting point out and the rod springs around until the strain is relieved, the action being repeated uniformly and indefinitely. The depth to which the cutting point enters the core depends upon its size, shape, and hardness, and upon the hardness of the core, also upon the intensity of the vibration. The distance through which the cutter rotates between points of contact with the core determines the number of threads to the turn, but it is probable that the number once established predisposes its regularity and symmetry. With lower speeds of rotation

the vibration is less intense and the cutting point remains in contact with the core for a longer period; the number of threads to the turn as well as the pitch of the threads may be changed in this manner.

The distance between threads of like number measured circumferentially is constant, but by increasing the period of contact the pitch may be changed without altering the number (see Nos. 2 and 3, Fig. 1 and Figs. 3 and 4). The greater the period of contact, the greater is the distance between threads measured at right angles, thus producing lack of parallelism between threads (see Figs. 3 and 4). It may be said, then, that the greater the actual distance between threads the greater the pitch and, conversely, the less the distance the less the pitch.

There is nothing to indicate that rate of advance affects the formation of riflings or threads on cores except in so far as it may limit the length of core affected; nor does the size of the core have any appreciable effect (see Fig. 7).

The length of the line of rods should and may have some influence in so much as the torsional vibration is probably affected by it. However, no appreciable difference in vibration was noted in the work done, as the tool approached the chuck which held the test piece, the vibration maintaining its intensity until the tool reached the chuck.

The conclusions reached by the investigations outlined above may be briefly summarized as follows:

1. Riflings of cores are produced by torsional vibration of the rods.
2. The formation of multiple threads, or a number to the turn, is determined by the intensity of vibration, character of the cutting medium and the core.
3. The pitch of the threads is determined by speed of rotation of the rods.
4. The size and length of rods probably act only indirectly to modify the size, shape, and pitch of threads.

As rifling of core is produced by rotation and varies in prominence with the intensity of vibration, it is possible to prevent or very materially reduce the vibratory action of the drilling mechanism by reducing the speed of rotation of the rods. This method of procedure would undoubtedly be much less difficult and troublesome and far more efficacious than the practice of greasing the line of rods.

It is possible that aside from the desirability of knowing how rifled cores are formed, the knowledge may be of little or no importance, yet who can say when a purely scientific fact may not become of considerable economic value.

The writer is indebted to many persons engaged in the manufacture and operation of diamond drills for the interest shown in the investigations and the information freely given, and wishes in this connection to express his appreciation of the assistance rendered.

Discussion

H. M. Roberts, Minneapolis, Minn. (communication to the Secretary*).—The rifling of drill cores is a frequent source of interest to men engaged in diamond drilling. Previous to the appearance of Dr. Crane's paper, there has been little record of connected observation or rigid speculation as applied to the cause of rifling. Replying to one of Dr. Crane's letters of inquiry, I ventured my opinion that the cause of rifling was a complex problem in physics, which I had not attempted to solve up to that time, except to ascribe it to the general cause of rotation and vibration of the rods. The question is worth attention not only as a matter of curious scientific interest but also for the reason that it stimulates thought on the mechanical action which takes place at the end of a diamond bit. In discussing Dr. Crane's paper, I still recognize the complexity of the problem and merely note a few observations of fact with the inferences which may be drawn from them. I am indebted for suggestions from W. J. Mead, F. F. Fredlund and James W. Hunter among others on the staff of the E. J. Longyear Co.

It seems clear that the cutting medium which causes uniform threading must be the diamond bit itself. The deep, regular rifling of such dense rocks as granite and norite for long intervals admits of no other reasonable assumption.

Rifling occurs in many different rocks, both hard and soft, but in all instances noted there is one feature in common: the rock is homogeneous over the extent of the rifling.

Wall of Drill Hole Rifled

The wall of the drill hole itself is rifled, as well as the core. This has been observed in the shaft of the Isabella Mine in Northern Michigan, which was sunk on a 2-in. hole in diabase.

Fig. 1 shows a specimen taken from the walls of this hole. It is reasonable to suppose, therefore, that rifling of the core is also accompanied in most instances by rifling of the wall of the hole. This leads to the inference that perhaps the immediate end of the bit is responsible.

The Diamond Bit

A consideration of the great pressures which bear upon a bit during the drilling makes it quite certain that all the stones work and engage the rock together. Examination of a worn bit shows that all the stones have played their part. In this connection it is of interest to note that

* Received June 24, 1916

while diamond setters place outside stones very accurately to gage, they seldom use a gage in setting inside stones, which perhaps admits of one stone working alone on the inside for some interval of time. Those portions of the stones which project from the immediate interior and

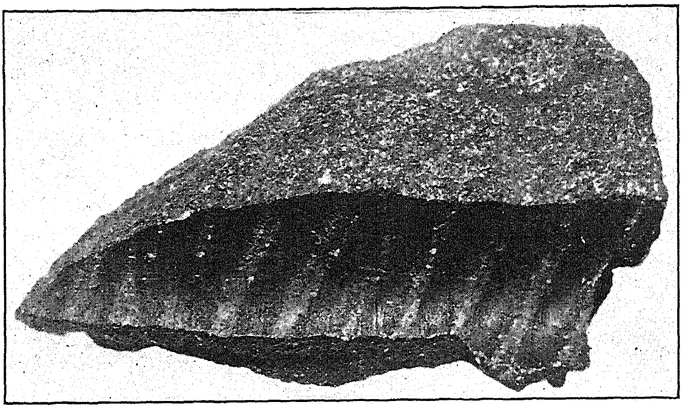


FIG. 1.—SPECIMEN FROM SHAFT OF ISABELLA MINE, CASCADE RANGE, MICH., SHOWING RIFLING ON THE WALL OF AN "N" DRILL HOLE.

exterior edges of the face of the bit are no doubt responsible for the threading. It is apparent that any play in the bit would act with greatest effect at the extreme edges and would permit of greater penetration than the overset of that portion of the stone which extends up from the

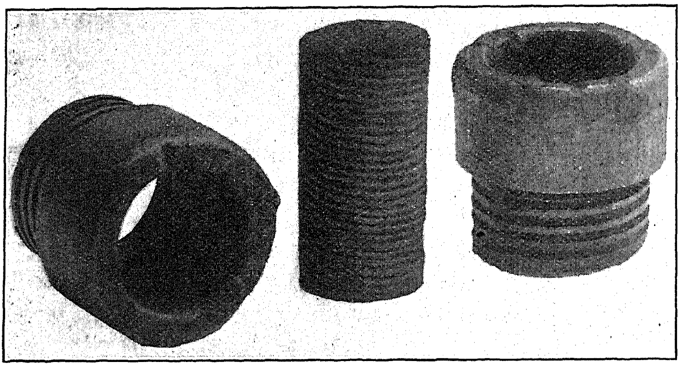


FIG. 2.—WORN DIAMOND BITS WHICH HAVE PRODUCED RIFLED CORE AND A SPECIMEN OF QUARTZITE CORE SHOWING LEFT-HAND THREADS. THIS CORE WAS PRODUCED BY A BIT REVOLVING TO THE RIGHT.

edge on the inside and outside of the bit. During the drilling operation the protuberance of the stones at the edges is much greater than when first set up, owing to the wearing away of the metal. Fig. 2 shows two worn diamond bits which have produced rifled core. There is no evidence to

show that a bit which has produced rifling was different in any material respect when first introduced to the hole than a bit which has produced smooth core. There is decided evidence to the effect that carbon wear and loss of metal are increased in any bit which has produced rifling. Thus the cause of rifling must lie in some force which acted upon the bit during the time when the rifled core was produced and which did not act during the time when the smooth core was produced.

Different Kinds of Threads and Varying Conditions

The depth of the hole does not seem to be a governing factor. Deeply rifled cores have been found in norite drilled with a 2-in. bit 10 ft. from the machine. Threads of different size, different pitch, and of varying numbers to the turn are to be found on cores of the same material from one hole drilled with the same bit and with the machine making the same number of revolutions per inch of advance in the bit. The threads are both left hand and right hand, although as far as my observation goes right-hand threads predominate. Fig. 1, showing the rifled wall of a drill hole, reveals both left-hand and right-hand threads in the same specimen. The bit producing these threads revolved to the right. I have counted both even and odd numbers of threads, varying from three to twenty, after which they become too fine to count; in fact, a smooth core always shows small variations from a true cylindrical shape.

Vibration of Rods Prevalent When Rifled Core is Produced

It is an observed fact that vibration of the rods is prevalent in drilling formations where regular rifling occurs. This obvious relation led to the replies which drill operators gave to Dr. Crane's question. The production of threads on a vibrating pipe in a lathe bears a distinct analogy and renders the relation between the vibration of the drill rods and rifling of the core quite certain. Let us examine this relation further, since we are pursuing the inquiry largely as a matter of pure interest.

Harmonic Motion of the Bit

The presence of threads over any interval of core indicates a constant play to and fro of the bit relative to the axis of the hole during its advance. Uniformity of the threading over any interval of core indicates that the swing of the bit operates according to some law. The fact that the number of threads and their character change from interval to interval, indicates that the factors causing the swing of the bit vary in considerable degree. We are evidently dealing with some type of harmonic motion. Possibly the homogeneous character of the rock found in most

rifled cores is one of the principal factors which permits of harmonic motion in the bit. In considering the instance where many threads of low pitch appear on a cross-section of core, it is possible at first sight to conceive that the threads are due to spiraling which is directly proportional to the advance of the bit, but in an instance where corrugations of steep pitch are produced, as No. 8, Fig. 1, of Dr. Crane's paper, with perhaps 600 revolutions of the bit to an advance of 1 in., it is evident that the process is extremely complicated. The production of left-hand core by a bit revolving to the right is also a difficult matter to explain (see Fig. 2).

Consider the forces that operate on a flexible line of drill rods which are revolving rapidly. First, compression; second, torque. The play of these two forces sets up waves which have their lengths parallel but twisted with respect to the drill rods and with their crests and troughs at right angles to the axis of the hole. The twist in the rods is perhaps not as great as might be supposed, for when a bit is blocked under great pressure, and the engine is stopped and released, the chuck seldom revolves back more than a turn or a turn and a half. During this process the bit is presumably fast on bottom.

As the rods revolve, transverse waves advance down the rods at every azimuth, like waves of light. That waves of this type are produced is shown by the whipping out of uncased holes when drilling soft iron formation. The samples are often vitiated in this manner. Anyone who will climb the tripod and look down on the water-swivel when the rods are vibrating will see there a distorted but nevertheless fairly true picture of what is taking place at the bit on the other end of the rods. It is possible to count distinct beats in the play of the swivel to and fro as the rods vibrate.

The regularity of the grooving in the threaded core indicates that the waves producing it have a definite time interval with a fixed relation to the revolution of the rod. The fact that all the stones are working at the same time does not present any particular difficulty when it is remembered that in any one instance of threading the stones are always at fixed intervals with respect to the axis of the bit and therefore have a fixed relation to any wave that affects the bit and a fixed relation to the period of revolution. In the instance cited by Dr. Crane where pentagonal core is formed, the time of vibration must be nearly a definite divisor of the time of revolution. That is, during one revolution of the bit there were approximately five major wave motions which produced a portion of five threads on the core for a longitudinal distance of less than $\frac{1}{100}$ in. In all other instances where the pitch is flatter the ratio must be more intricate. When right-handed threads are produced, the period of vibration is incommensurable and in this instance thereby deferred slightly each time with respect to the core as the rod revolves. When left-

handed threads are produced, the period of vibration is also incommensurable, but in this case occurs just an instant earlier with respect to the core during each succeeding revolution.

Overtones

In wave motions of this kind it is conceivable that there are overtones; that is, if the period of one wave length be T, then there are wave lengths of $\frac{1}{2}\ T$, $\frac{1}{3}\ T$, etc. The braided threads on many pieces of core are suggestive of secondary action of this kind (see Fig. 2 of Dr. Crane's paper). All of the laws of resonance and interference in wave motion would enter into the amount of play in the bit as it revolves. Again, when relatively smooth core is produced, it is probably the result of the impact of countless numbers of periodic waves offsetting each other.

There are many other apparently simple phenomena in nature which are due to a combination of periodicities, as a beam of light or the general strike of a complexly folded iron formation.

The net result of the wave motions down the rods operating under the laws of resonance is translated into a wave whose length is nearly parallel to the circumference of the core and whose amplitude is along the radius, thus producing uniform threading. This is recorded in the rock like the mark on the indicator card of an engine. Only in rare instances do all the factors combine under the laws of resonance so that a deep definite tracing of the wave motion is formed.

Mathematical Interpretation

We may attempt to arrive at a rough mathematical interpretation of wave motion at the end of a diamond bit as expressed by threads in the core.

Let R = the number of revolutions of the bit during any interval of advance.

N = the number of threads appearing on the cross-section of core

$F\ (p)$ = some continuous or discontinuous function of the pitch of the threads, p having values from negative to positive infinity, $F\ (p)$ approaching the values -1 and $+1$ at these ends, and approaching the value zero as p approaches zero, the value of p being related to the value of N.

Let W = the number of major wave motions during any interval of advance of the bit, as indicated by the threads.

Then $W = R\ N\ F\ (p)$

Thus in the instance where pentagonal core is formed, and N equals 5, p approaches infinity and $F\ (p)$ approaches the value 1, if it be assumed

that 600 revolutions of the bit have been made during the delivery of 1 in. of core; then $W = 5 \times 600$ or 3,000, which is indicative of the number of major vibrations of the bit with respect to the axis of the core during its spiral advance of 1 in.

When the pitch p is left-handed, $F(p)$ may be considered as varying from 0 to -1; when the pitch is right-handed $F(p)$ varies from 0 to $+1$. As p becomes small, in either case, the value of N increases and W increases approaching the case where many fine threads of low pitch are produced; W increases greatly, which is to say that a cylindrical core is formed when there is a large number of exceedingly small wave motions of the bit to and fro. Considering an instance when N remains constant and the threads ultimately coalesce while "p" decreases in value, then the value of W decreases. The ultimate end of this process is the production of a smooth core with slight wave motion of the bit.

Inspection of this equation shows that as R increases, W must increase. This expresses the violence of vibration when harmonic motion of the resonant type is set up during a high speed of revolution.

This mathematical statement is merely an attempt to express some of the relationships which appear, by speculating on the nature of the phenomenon. It is doubtful whether it is possible to determine rigidly the value of $F(p)$ in any instance or to tell why p should be left-handed or right-handed or to devise any means of telling why N is 5 in one case or 9 in another. The strength of the rods, the number and character of the carbon, the way they are set with respect to each other, the water pressure in the hole as determined by the force of the pump, the clearance of the bit, etc., the amount and character of the deformation in the metal of the bit at any particular time, the kind of drilling machine, and the manner of its set up, the depth drilled, the nature of the rock, the presence of soft horizons above in the hole which may affect the vibration of the rods, the speed of rotation and the pressure on the rods—the factors which influence these values are numerous and therefore difficult to analyze into their true proportions.

Practical Considerations

Dr. Crane's practical application of the results of his research is discouraging in these days of keen competition for drilling contracts. I refer to the suggestion that it is possible to reduce "the vibratory action of the drilling mechanism by reducing the speed of rotation of the rods." An old drillman in commenting on this says that he can break up the vibration by increasing the speed of rotation. This suggestion might better be stated thus: The violence of vibration can be broken up by varying the speed of rotation.

This destroys one of the constant conditions essential to harmonic

motion. The greasing of the rods dampens the wave and sets up interference by releasing friction at antinodes of the waves, thus destroying another constant. This may be inferred by examining the line of greased rods when pulled from the hole and by observing that the grease is worn clean at definite intervals.

In making this comment on Dr. Crane's conclusion, I would not be understood as casting doubt on the utility of this type of research. An analysis of all the conditions which govern the advance of a diamond bit might enable operators to control the direction of diamond-drill holes. At present the hole usually takes its own direction.

Some practical considerations have occurred to me while dealing with the subject. They are no doubt commonplace to many old drillmen, but seem worth recording: When drilling deep holes in homogeneous formations, it will reduce the carbon wear and increase the footage to use stiff new rods and to use a blank bit which has the smallest possible excess in diameter over the size of the rods. The use of eight stones in a bit rather than four or six will reduce the possibility of one stone carrying the burden of the work. The bit should be reset frequently so that deformation of the weakened steel in long-used bits will be avoided; thus the gage of the hole may be kept more accurately. These precautions will set up conditions adverse to the development of resonant wave motion to and fro in a bit and thus tend to keep it working straight ahead, which is the true business of a diamond bit.

Method of Mining Talc

BY F. R. HEWITT, ASHEVILLE, N. C.

(Arizona Meeting, September, 1916)

THE methods of mining talc are simple, and in western North Carolina are almost entirely by open cut and quarry. The larger part of the talc of this section lies in various-sized "veins" inclosed in quartzitic walls, the majority of which have by folding been thrown into a perpendicular position, or nearly so, and the exposed edges of which have been covered by débris from erosion of the mountain above. This covering of loose rocks and earth is from 5 to 25 ft. thick, and in most cases has to be removed because it is difficult to support. Sometimes the covering is heavy enough to "catch" and timber successfully, and in this case the talc can be worked out by the usual method of following the "vein," using stulls for holding up the walls until the vein is worked out, and then allowing the cut or drift to fall in.

In some cases it proves more convenient to sink shafts and run drifts, following the talc "veins" until exhausted. The talc deposits of this section are badly broken and faulted both laterally and perpendicularly, and in some cases are found many feet below the level of drainage, necessitating pumping at considerable expense.

My own experience in mining talc has been that the chief point is to get a reliable "vein," and that then the mining is not as difficult, or any more so at least, than that of any common ore. If the "veins" are pure talc it is simple; if the talc is admixed with foreign matter, such as tremolite in excess, or stains of iron or manganese, as is often the case, one had better abandon the deposit.

Methods of Manufacture

Talc, being a soft mineral, is not difficult to reduce to the condition desired by the trade. Powdered talc is admitted under low duty from France, Italy and Austria, and some is being imported from Asiatic points. Talc is usually ground and bolted in any simple reduction mill. That mined in this section is hand-sorted and the quality that will make good powdered talc is ground for use in the manufacture of talcum powder, cosmetics, etc., while all that is hard enough and has sufficient strength is used for crayons and blanks for gas burners and electrical work, being sawed into the shapes desired. The rest that is clean and pure is ground

in mills, of which many different kinds are used. Some people like the mill stone, while others prefer the many kinds of high-speed beater mills for grinding.

The manufacturing process can be summed up as follows:

More or less hand-sorting may be necessary, and in grinding, the talc must be reduced sufficiently to pass through a 170- to 200-mesh sieve, silk cloth reels being commonly used.

Among the objectionable impurities which it is almost impossible to remove are lumps of tremolite and pyrite. They destroy the value of the talc by changing the color and making a coarse and hard gritty product which is not saleable.

Uses

Most of the talc sold is used in the paper, rubber and paint trade, while the lower grades go largely into roofing. About one-quarter of the production goes into talcum powder. Not a little is employed as a body or carrier of medicinal chemicals used in tablet form.

Much of the talc or soapstone is cut with small saws into crayons of various sizes for use in the iron-working trades and on blackboards. A growing business is the manufacture of gas burners and small blocks for electrical work, which, after shaping and burning, are known as lava goods.

The two last, crayon and burner manufacture, are the most important uses of talc, but the greatest difficulty is to find mineral of the proper quality, as it has to be solid, firm, and free from grit and other foreign matter.

Stoping in the Calumet and Arizona Mines, Bisbee, Ariz.

BY PHILIP D. WILSON,* B. S , E. M., WARREN, ARIZ.

(Arizona Meeting, September, 1916)

THE mines of the Calumet & Arizona Mining Co. are situated in the Warren Mining District, Cochise County, Arizona, between Bisbee and Warren and adjoin those of the Copper Queen Consolidated Mining Co. and the Shattuck Arizona Copper Co. Paleozoic limestones of considerable thickness have been intruded by granite porphyry. A great stock of the intrusive outcrops boldly near the city of Bisbee and is known as Sacramento Hill. From this central core the porphyry has tongued out into the limestone for many thousands of feet as dikes and sills of irregular shape and variable dimensions. Churn drilling has recently developed a large tonnage of secondarily enriched copper ore in the main porphyry mass of Sacramento Hill. While some ore has been found in porphyry in other parts of the district, it is relatively unimportant compared with that which occurs in the Paleozoic limestone as irregular lenses replacing favorable beds near porphyry contacts or associated with well-marked fracture zones often many hundreds of feet away from the intrusive.

In a zone extending from Sacramento Hill for a distance of from 1,500 to 3,000 ft., the limestone has been extensively metamorphosed and in this contact metamorphic zone the ore occurs through a known vertical thickness of upward of 900 ft. and individual orebodies often show extension in a vertical direction of over 200 ft. At a greater distance from the intrusive stock, the ore occurs as lenses of greater horizontal than vertical dimensions with an average thickness of approximately 30 ft. An exception to this rule is found in the occasional steeply dipping mineralized fractures, almost true tabular veins, which are found long distances away from the zone of contact metamorphism.

The copper was probably introduced during or slightly later than the porphyry intrusion in the form of chalcopyrite and bornite. These primary copper minerals are always associated with pyrite and often occur as rich lenses in a huge mass or shell of pyrite carrying an insignificant percentage of copper. In many instances the copper-bearing solutions were preceded by siliceous solutions and these latter, often persisting during the period of mineralization, have thoroughly indurated the ore-

* Assistant Geologist, Calumet & Arizona Mining Co.

body and the surrounding country rock. Oxidation took place during two distinct periods separated by a period of submergence in the Cretaceous sea and subsequent elevation, distortion and tilting of the sediments. This has resulted in a most irregular zone of oxidation. In some cases enriched chalcocite ore and even thoroughly oxidized ore are found stratigraphically far below primary orebodies. Where oxidation has been complete the carbonates and oxides of copper occur beneath or entirely surrounded by a mass of oxides of iron and aluminum, often of great extent. While there has been considerable enrichment with development of chalcocite in the district, opinion, in view of recent developments of the primary ores, is tending toward the conclusion that migration of values has been for the most part local and that both the oxidized and the chalcocite ores depend for their present copper content rather on rich primary sulphides than on any extensive leaching and subsequent enrichment of low-grade material. The large masses of iron and aluminum oxides found in connection with the oxidized ores are then the oxidation products of the shells of lean pyrite and pyritic limestone which surround the primary copper sulphides rather than gossans from which appreciable copper has been leached.

Conditions Determining Stoping Methods

The great variety of ore occurrences in the district, and the equally variable character of the country rock in the vicinity of the orebodies, render any uniformity of stoping method impossible where the factors of economy and efficiency are considered. In the early days of Bisbee all of the ore mined was taken from the irregular high-grade oxidized bodies found within a comparatively short distance of the surface. Square-set stoping with subsequent filling was used with much success and even today this time-tried method of Philip Deidesheimer is recognized as the system most widely applicable to the Bisbee ores. It was not until comparatively recent years that the natural reluctance to try new methods while the old could still be used was overcome, in the Calumet and Arizona mines under the superintendence of W. B. Gohring, with resultant reduction of costs in some cases to less than one-half of those formerly obtained.

The considerations determining the stoping method to be used may be summarized as follows:

1. Safety and efficiency of working conditions for the men.
   (*a*) Physical characteristics of ore and country rock.
   (*b*) Ventilation.
2. Maximum ultimate profit.
   (*a*) Cost of mining.
   (*b*) Ore which may be economically sacrificed.
   (*c*) Relative economy of mining and of sorting out included waste.

(*d*) Rapidity with which it is advisable to mine, or daily tonnage required.

3. Shape and dimensions of orebody.
4. Safety of mine.
   (*a*) Cost of timber.
   (*b*) Availability of filling.
   (*c*) Fire risk.

Square-set Stoping

The square-set system is easily the most flexible and where the orebody is very irregular and has large included blocks of waste it is the most satisfactory. It is a simple matter to leave the waste behind as a portion of the filling, and where the mining of too large a section at once is not attempted, and filling is kept within a reasonable distance of the back, it is as safe or safer than any of the other methods in vogue. By this method the cost* of stoping per ton including labor, powder, timber, carbide and air, ranges from $0.80 in sulphide ore to $1 or even $1.30 in oxide ore. Where the ground is very heavy, additional timber as bulkheads and double-up sets increases the cost materially and in sulphide ore the forest of timber makes the risk of fire an important consideration. In a normal square-set stope the item of timber, which laid down at the mine costs about $17.50 per thousand, amounts to 25 per cent. of the total.

Recovering Square-set Stope Timbers

The first attempt to reduce this expense was made some years ago by M. W. Mitchell, foreman of one division of the Calumet and Arizona mines, who devised a method for "robbing" a considerable portion of the timber from a square-set stope during the process of filling. Approximately 50 per cent. of the timber can be recovered where conditions will permit of this robbing process, and the actual stoping cost based on a series of stopes over a considerable period in which the timber has been recovered and used again is reduced by about 8 per cent. While applicable to a large majority of square-set stopes, in many instances the ground is too heavy to risk removing the timbers and in some cases the timber is so badly damaged that it is not worth extracting. The method finds its greatest economy where an orebody is being mined in successive sections so that the recovered timber may be left in open square sets at the edge of the stope and used when the adjacent section is being worked, as the cost of excessive handling soon reduces the saving to a minimum.

When an attempt to rob the timbers from a square set is anticipated, the first section is carried up five sets wide, as shown in Fig. 1. The

* These costs obtain under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers. During the past year the sliding wage scale and the abnormally high cost of supplies have resulted in a material increase in all mining costs.

length of the section is determined by the character of the ground and varies from five to as many as ten sets. It is advantageous to carry the section as long as the ground will permit, since the longer the section the greater will be the ultimate timber recovery. It is usually not feasible to attempt to recover timber from a stope over 50 ft. in height. When an orebody, the thickness of which is greater than this, is to be mined and the timber recovered by this method, the operation is carried on in successive lifts, as will be hereinafter described. In stoping the section a gangway is maintained in the central row of sets on the sill floor and chutes, *ee*, built in each alternate set on either side. Slides at appropriate places in

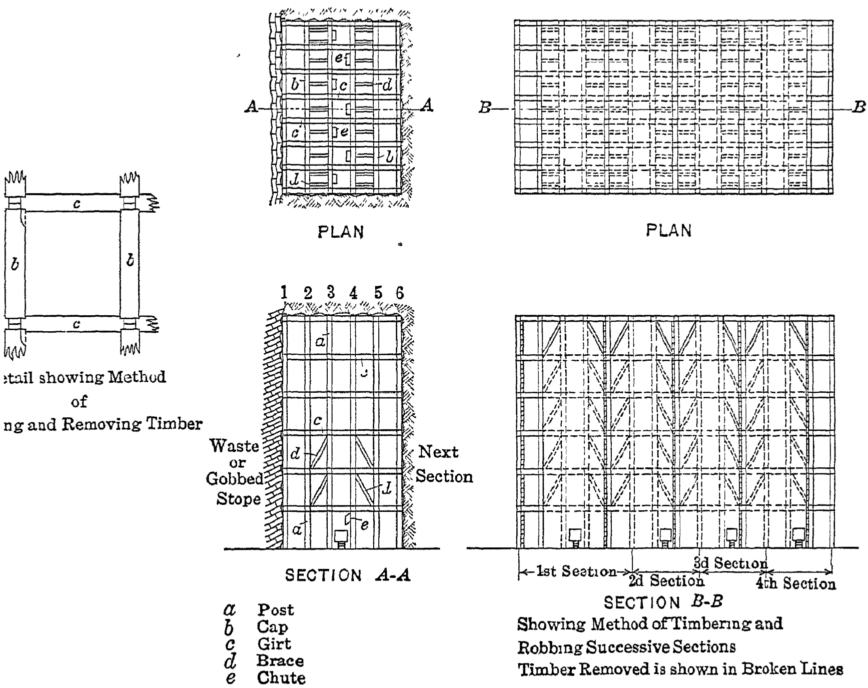


FIG. 1.—MITCHELL METHOD OF RECOVERING SQUARE-SET STOPE TIMBERS.

the stope will deliver the broken ore to these chutes as stoping proceeds, with a minimum of shoveling. The square sets are so erected that the caps, *bb*, lie parallel to the gangway and the girts, *cc*, consequently at right angles to it. When the ore has all been extracted and the stope is ready for robbing, 6 by 6-in. braces, *dd*, are placed between the two diagonally opposite caps in the second and fourth rows of sets on the two floors above the sill, as shown in Section *AA*, Fig. 1. If the stope is taking weight badly a brace may be necessary at each end of each of these caps, but in some cases a single brace in the middle of the cap is sufficient. It is usually the case where robbing is attempted that the plans include the

robbing of successive sections which will each be mined in turn after the preceding one has been robbed and filled. If such is the case, gob lagging is spiked to the rows of posts 1 and 5, Section *AA*, so that when filling is introduced the gob will lie tightly against the ground or previously gobbed stope on that side of the section, in this case the left, where no further mining is anticipated. One run of sets on the other side of the section will be left open when the stope is filled, so that the timber as extracted may be stored in these sets and used when the adjacent section is carried up.

After the braces have been put in place the caps in row 2 are cut, as shown in the detail, Fig. 1, to permit of slipping out the girts between rows 2 and 3. When these have been removed it is a simple matter to remove the caps and the sill and first floor posts in rows 3 and 4 and the girts between rows 3 and 5. Waste filling is then introduced until it reaches to within a few inches of the second floor above the sill, the caps cut and the timber on this floor between rows 2 and 5 removed as before. The braces, partially buried in the gob, may be drawn out by throwing one end of an ordinary mule skinner's tail-chain over the exposed end of a brace and fastening the other to the end of a piece of timber. Using this piece of timber as a lever and the cap still in place above as a fulcrum the brace is easily loosened and drawn from the filling. During each downward movement of the lever the chain automatically loosens and slips lower on the brace, so that one man can easily manage the operation. The braces as removed are placed diagonally between the caps at the top and bottom of the third floor above the sill. These braces may obviously be used again and again until weakness or damage impairs their usefulness. The process as described with alternate filling and drawing timbers a floor at a time is repeated until the top of the section is reached and the stope filled. The timber as removed is stored in the open run of square sets for use in the adjacent section. In all cases the posts and girts on each end of the stope are left behind and if the ground is unusually heavy it is safer to leave an entire run of sets to support the ends of the stope as the timber in the center is being removed. These are left in the gob so that the ultimate timber recovery is much reduced. The braces, as the timbers below are removed, take up the vertical and part of the lateral pressure, but the major portion of the latter is taken up by the gob. It is essential, therefore, that the filling be kept well up to the level of the floor from which the timber is being removed.

If the ground is unusually heavy or the orebody over 50 ft. in thickness the section must be carried up in successive horizontal lifts in order that there may be a minimum of ground open at any one time before filling is introduced. It is the usual practice to open the section three sets high, brace, remove the timbers and fill as before the lower two floors. With the stope thus securely reinforced, the section may be carried two

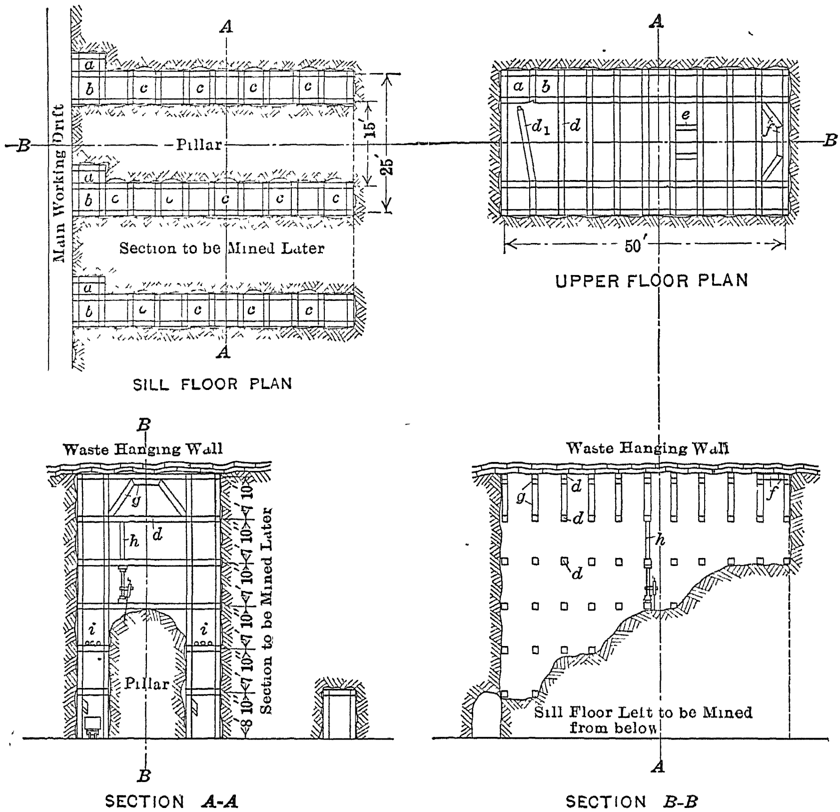
sets higher when the two lowest open floors are robbed and filled as before. If the ground is so heavy that additional precautions are necessary the timber is first drawn from only one end of the stope and filling is introduced into the robbed portion while the adjacent row of timber on the same floor is being extracted. Thus, as soon as the timber is removed, it is replaced by gob so that the walls of the stope are at all times amply supported. Using this method of mining and recovering the timbers in successive lifts, there is virtually no limit to the height to which a section may be carried safely in heavy ground.

The adjacent section and each subsequent one are carried only three sets wide, but the same length as before. The method of timbering and robbing successive sections is shown in Section *BB*, Fig. 1. In each case the stope is carried up alongside of the run of square sets left open in the previously mined section so that four sets of ground are actually open. The timber extracted from the preceding stope is used as far as it goes in the new section. That which is in the best condition is used where it can be recovered again and the partly damaged material where it will be left behind in the gob. When the robbing process begins in a new section the braces are placed in the run of sets left open in the previously mined section and the center run of the new one. The caps cut are those directly against the gob. In every case it is the caps to be left behind that are cut, leaving unmutilated all of the timber to be removed. As filling is introduced, gob lagging is spiked as shown to the outside of the center run of sets, leaving one run open as before for storing timber for the next section. It is evident from Section *BB*, Fig. 1, that in using this method in several successive stope sections only every third row of posts and caps is left behind in the gob while all the intermediate timbers are recovered. The secrets of success in using this method are the narrow stopes and care in keeping the filling well up to the floor from which the timber is being taken.

Mitchell Slicing System

Mr. Mitchell is also responsible for the next improvement in stoping methods which has found wide application in the Calumet and Arizona mines during the past few years. This method, known as the Mitchell slicing system, has been described in an article by M. J. Elsing in the *Engineering and Mining Journal*, July 23, 1910, but as several changes and improvements have been made in it since that time it will bear further mention. It was arrived at more or less by accident. A large block of ore broke away from the back while a heavy sulphide stope was being worked, and in order to recover it, long stringers were thrown across the top of the ore to support the back and the ore mined from above by underhand stoping. The method as finally developed from this simple beginning is applicable to orebodies in which the hanging wall is fairly

lat and regular and the lateral pressure not too great. It is flexible, however, by virtue of the fact that square sets may be used in conjunction with the method in mining irregular and outlying portions of the ore-bodies. Furthermore, there must be no large quantity of waste in the ore, for while it is possible to sort and leave some waste material behind in



a Manway
b Main Chute
c Auxiliary Chutes
d Stringer in Position
d_1 Stringer being Swung in Position
e Temporary Spread
f Diamond Brace
g Segment Set
h Temporary Brace
i Grizzly

FIG. 2.—MITCHELL SLICING METHOD.

he stope the method loses many of its advantages if careful sorting must be done.

After preliminary prospecting has roughly outlined the shape and dimensions of the ore shoot, two lead rows of sill square sets are driven, 20 ft. center to center, at right angles to the general strike of the body from the main working drift to the end of the section to be mined, as shown in the sill floor plan, Fig. 2. The length of the section varies from 25 to 50

ft., depending upon the character of the ground. Where it is possible to run cars directly into these two crosscuts a small sloping cut is made in every alternate set on the same side of each crosscut for a loading chute. Care must be taken to keep these cuts small and under no circumstances to take them on both sides of the same pillar, for if the base of the pillar is weakened trouble is sure to ensue. If it is not practicable to load the cars in these side crosscuts, as is often the case, the first set alone may serve as a chute and the cars be loaded directly in the main drift. A six-post raise is carried over the first two sets in each cross-cut through to the level above to be used as a timber and manway and later as a waste chute. On the sill floor a small manway, *a*, is cut out next to the chute set as shown, and on the floor above the ladderway is carried up over the chute while the other compartment of the raise is lined for a chute. When the raises have holed to the next level runs of square sets are carried directly over the lead rows of sill sets to the top of the ore, leaving a pillar of ore 15 ft. wide and the length of the section resting on its own base to be subsequently mined in horizontal slices from the top downward. Under normal conditions standard 10 by 10-in. square-set timbers are used in these preliminary operations, but if the ground is unusually heavy 12 by 12-in. timbers are advisable.

The first slice is taken, as shown in Sections *AA* and *BB*, Fig. 2, over the whole top of the pillar, cutting it loose from the waste hanging wall. Machines are first erected in the topmost square sets and holes pointed in such a manner that the ore passes with a minimum of shoveling onto grizzlies in the open run of square sets. Where only one chute is used the broken ore fills the sets until its normal angle of repose is reached, the remainder running from the chute by gravity. The 10 by 10-in. stringers, *dd*, 15 ft. long and framed with 2-in. tenons, are thrown across the open slice between the caps of the topmost run of square sets and directly under the hanging wall with such blocking and lagging as is necessary to keep the stope safe and the timbers secure. In placing a stringer in position the tenon on one end is inserted in the place between caps and post framed to receive it in the same manner as an ordinary girt. The opposite cap is cut 2 in. to allow the tenon on the other end of the stringer to be swung into position, as shown in the upper floor plan, Fig. 2, and a piece of lagging spiked to the cap to hold it in place. The topmost stringers are supported temporarily by stulls resting on ore in the pillar until enough ground has been broken to permit of laying similar stringers between the caps of the next lower run of square sets. When these stringers, resting on the unbroken ore in the pillar, have been placed in position the stulls are replaced by segment sets, *gg*, of 10 by 10-in. timber, erected, as shown in Section *AA*, Fig. 2, to support the upper stringers and the back of the stope. While 10 by 10-in. or 8 by 10-in. stringers are necessary directly under the hanging wall, it is the usual

practice to use 8 by 8-in. timber for those subsequently put in where the weight to be carried is much less. Segment sets are usually unnecessary except below the topmost stringers, although occasionally used below if the ground is very heavy.

The next slice is then taken across the top of the pillar. Either jack-hamers or piston machines set up on columns between the stringers are used to drill down holes, pointed so that the ore is broken directly into the chutes. In subsequent slices, if a piston machine is used, it is usually necessary to put in temporary 6 by 6-in. braces, *hh*, between the stringers vertically above the drill column in order to make the machine secure while drilling. It is sometimes safer to put in temporary horizontal spreads, *ee*, between the stringers near the holes which are about to be blasted. If the ground at the end of the stope begins to swell or work 10 by 10-in. diamond braces, *ff*, are placed in position as shown in the upper floor plan, Fig. 2, to take up longitudinal pressure. The slices are taken to within one floor of the sill, leaving one set of ore to be mined from below. A floor is then laid over the entire stope and 2-in. gob lagging is spiked vertically to the inside of both vertical runs of square sets, leaving the square sets open when the stope has been filled to be used as one set of chutes in each of the adjoining sections. If the ground is very heavy it has been found impracticable to leave these sets open, and in that case they are lagged on the outside and filled with the rest of the stope. The filling is introduced through the two raises from the level above and if the stope is not taking weight badly a considerable proportion of the stringers and segment sets, if the latter have been used, may be recovered as filling progresses. Upward of 50 per cent. of the stringers may be thus removed under favorable conditions. If it has been considered advisable to leave the square sets open the timber recovered may be stored in these sets and used when the adjoining section is taken. Where the timber cannot be used in a stope in the immediate vicinity it is usually better economy not to attempt to extract it, as much handling soon neutralizes any saving gained.

The maximum vertical height of a section which may be safely mined by this method has been found to be about 60 ft. When the ore extends from level to level, or 100 ft. thick, the stope is worked in two lifts of 50 ft. each, the upper lift first. In this case raises are driven through from level to level as before but the preliminary rows of square sets are driven 50 ft. above the sill and carried from there to the upper level. An attempt was made at first to carry these runs of sets the entire 100 ft., but it was found that by the time the upper lift was mined out and filled the timbers below were so badly crushed as to be useless. The upper lift is mined as has been described with the exception that the slices are taken for the entire 50 ft. before the floor is laid and filling commences. It is obvious, too, that all of the ore must perforce be drawn from only two

chutes instead of from the series of chutes as is optional in the method as originally devised.

Where waste material must be sorted from the ore and left behind, flooring may be laid on the stringers and a certain amount of waste piled on the floors. A recent adaptation of the method, which eliminates to a certain extent this makeshift device, depends on mining the pillars in benches as shown in Section *BB*, Fig. 2. The slices are taken out more rapidly in the end of the stope nearest the waste raises so that the lowest slice is mined out, the floor laid and filling brought in while mining is still progressing on the upper benches at the other end of the stope. When this stage is reached any waste encountered may be mixed with the filling in the worked-out end of the section. Mucking broken ore is further reduced by this improvement, as the ore runs more readily into the chutes if the benches are properly taken.

In preparing to work a section adjoining a filled stope, a new lead row of sill sets is driven, as shown in the sill floor plan, Fig. 2, leaving a 15-ft. pillar as before, the vertical run of square sets carried to the hanging wall, and this run of sets used in conjunction with the one of the other stope which is still open, if it has been possible to so leave it, as chutes in working the new section. Thus but one run of square sets with the consequent narrow work is necessary for each section. If it has not been possible to leave the square sets open in the filled section it becomes necessary to take the slices at such an angle that the broken ore will run by gravity into the new run of sets. The bench method is particularly useful in this regard, as it facilitates the gravity handling of the ore.

The saving in labor, timber, and powder by this method over the square-set stope is an obvious and substantial one, but the main advantage of the system is the rapidity with which the ore may be mined if necessary and the increased tonnage yield per man in the stope. While 5 or 6 tons per man is considered creditable in a square-set stope, it is not considered extraordinary in mining the pillars in the Mitchell system to reach three or four times those figures and in auger ground a yield as high as 50 tons per man per shift has been attained. The actual saving in stoping cost as shown in the appended table amounts to from 20 to 30 per cent. of that of square-set stoping.

Mitchell Top-Slice Caving System

It was decided about two years ago that the top-slice caving method as used on the Minnesota iron range, in Cananea, and elsewhere, might be applied to a large body of oxide ore in which heavy and swelling ground made the cost of square-set stoping excessively high. The fact that there was no working level above this ore from which the stopes might be filled also contributed to the choice of the top-slice system. Although

the method as usually practised worked with some degree of success and reduced the cost of stoping substantially, Mr. Mitchell, who was in charge of the work, was not satisfied with the results, and from it he evolved an inclined top-slice caving method which reduces the large amount of handling of ore in the stope, incidental to ordinary top-slicing, to a minimum.

The method is applicable to the same conditions as those under which

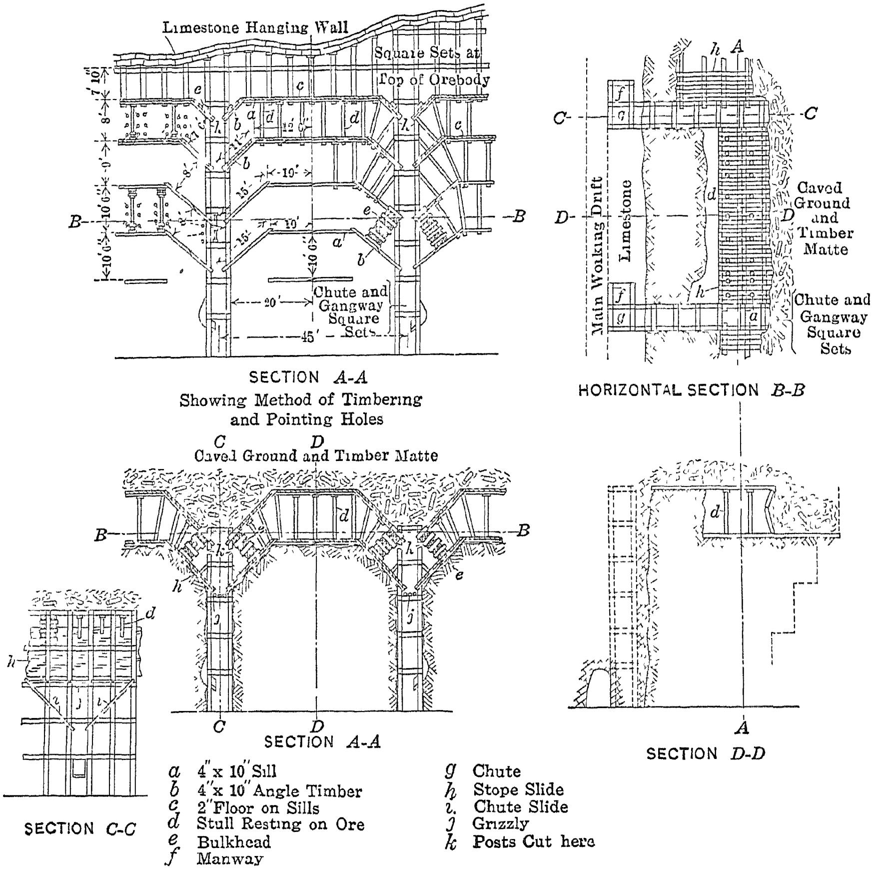


FIG. 3. MITCHELL TOP-SLICING CAVING SYSTEM.

ordinary top-slicing is successful, that is to say, uniform ground which is too heavy for economical square-set stoping and which will cave readily as required, but may be at all times kept under adequate control. It may often be used with increased economy in ground adapted to square-set stoping if the other conditions obtain. Before deciding upon this method there should be reasonable assurance that there is no ore left unextracted above the body to be caved, as the subsequent mining of this

ore would be attended by many difficulties after the country has been well broken up. In case development work is being done on a level above that of the stopes, this method, unlike square-setting, affords no means of disposing of the development waste, which consequently must be trammed to the shaft and hoisted. In common with ordinary top-slicing, ventilation is poor in the stopes worked by this system. There is sure to be some ore sacrificed in the process even under the most satisfactory conditions, which is an important consideration in the Warren District where a comparatively high-grade ore is mined.

In preparing for this method, the orebody is laid out in 45-ft. sections at right angles to its general strike. These sections may be mined the width of the ore to a maximum of about 80 ft., and 100 ft. is the economic limit of the height to which each section should be carried. Permanent six-post raises are driven, as shown in Sections *AA*, *BB* and *DD*, Fig. 3, preferably in waste at the edge of the ore, to the elevation of the top of the orebody. One compartment of each is used as a chute, *g*, the other as a timber and manway, *f*. From the chute compartment, parallel lead rows of square sets are carried the width of the ore, or of the section to be taken, on the three floors directly beneath the waste hanging wall. On each of the floors below these the square sets are carried a distance decreasing as the sill is approached leaving a foot wall of unbroken ground sloping in the direction of the main drift on which the broken ore will run directly into the chutes. This leaves a pillar 40 ft. wide which is mined by horizontal slices from the top downward, one-half from either side simultaneously, and the back allowed to cave as stoping progresses.

Before taking the first slice, the ground on either side of the top row of square sets is broken by holes drilled from a set-up in the square sets in such a way that 4 by 10-in. angle timbers, *bb*, 7 ft. 6 in. long, may be rested on solid ground at an angle of about 45°, as shown in Section *AA*, Fig. 3; 2-in. flooring is laid on these angles, making a slide, *h*, upon which the subsequently broken ore will run by gravity upon grizzlies, *jj*, in the square sets and thence into the chutes. The first cut is taken from both sides simultaneously across the pillar at the end of the ore or section farthest from the main drift and is timbered with standard square sets. The slice then proceeds from this cut toward the main drift, working from both sides of the section at once and supporting the waste back with square sets; 4 by 10-in. sills, *aa*, cut to lengths and with bevels as shown in Section *AA*, Fig. 3, are laid at the base of the posts, furnishing a support for the upper end of the angle timbers. Beneath the beveled joints of these sills pieces of 2-in. plank are spiked to render them rigid and to act as headboards for the stulls, *dd*, placed under them when the next slice is taken; 2-in. flooring is laid on top of these sills. As the slice retreats toward the main drift the rows of square-set posts first erected are shot out and the back caved, leaving at all times at least two open

sets between the working face and the caved ground. To prevent the vertical run of square sets from being crushed by the weight of the caved ground, small bulkheads, *ee*, are built under the lower caps of this floor resting upon the angle timbers. The posts are then cut at points, *kk*, directly under the angles, thus transferring the weight from the square sets below to the solid ground in the pillar.

When the upper slice has proceeded far enough from the extremity of the section so that the ground first caved is solid, preparations may be begun for the second slice. A machine is set up in the open run of square sets and ground broken so that 4 by 10-in. angle timbers 11 ft. long may be laid as before at a slope of about 45°. The angle timbers and sills above are caught up as shown with round stulls resting on the unbroken ore in the pillar, 6 to 8 in. in diameter as the ground may require. The preliminary cut is carried as before about 5 ft. wide and 9 ft. high across one-half the width of the section, or 20 ft., erecting stulls under the sills beneath the caved ground and timber matte above; 4 by 10-in. sills are so laid that they do not come vertically below the upper sills, for the vertical stulls supporting the upper sills should rest directly on solid ground. These sills are covered with 2-in. flooring as before. The erection of drill columns and the pointing of holes is shown in Section *AA*, Fig. 3. The breast holes shown are usually unnecessary, as the cut holes and lifters are in almost all cases sufficient to break the ground. After the preliminary cut has been taken the major portion of the ore is slabbed directly into the chute sets, leaving from one-third to one-fifth to be mucked. Stulls should be just strong enough so that the third stull back from the working face is commencing to crush as the face is being drilled, as shown in Section *DD*, Fig. 3.

As slicing proceeds downward the lower floors of the vertical runs of sets must be extended to maintain the slope of the solid ground to the extreme end of the section and permit the broken ore to run readily into the chutes. Care must be taken not to take out the vertical runs of sets too far ahead of operations, for the weight on them is considerable even if the greatest precautions are taken. If the orebody is over 80 or 100 ft. in width and consequently too wide to be mined in one section at right angles to its strike, it is not advisable to attempt to leave a vertical wall of ore against the caved ground and timber matte at the end of the section. In this case each new slice is started about 5 ft. closer to the working raises than the slice above, as shown in broken lines in Section *DD*, Fig. 3. Thus the end of the stope is left in steps and the danger of losing ore which may break from the wall and mix with the matte is eliminated. The ore so left may be recovered readily when the next section is taken. As mining approaches the sill floor, chutes may be put in the sill sets, as shown in Section *AA*, Fig. 3, at such intervals as will facilitate the rapid and efficient drawing of the broken ore. Bulkheads, with subsequent

cutting of posts to take the weight from the vertical run of sets, need not be erected oftener than conditions require. In normal operations bulkheads in every third slice afford sufficient protection to the square sets below. Enough partly crushed timber comes through from the timber matte above to build these bulkheads, so that their cost is insignificant. It has been found that the maximum thickness of slice which can be safely carried under the conditions encountered in the Calumet and Arizona mines is 10 ft. 6 in. and slices of this thickness are not attempted until the fourth slice is taken and the caving matte is under perfect control. If waste is encountered in the ore it is quite feasible to sort it out and leave it behind to become incorporated with the caved ground and timber matte, but like the method last described the system loses many of its advantages if careful sorting must be done.

As shown in the appended table, a 10 per cent. reduction in stoping cost is gained by this method over ordinary top-slicing. The labor cost is cut about 15 per cent. and the item of powder is substantially reduced. The ore may be mined rapidly and safely in the heaviest kind of ground. An output of about 10 tons per man per shift or a total of about 125 tons per day can be maintained from each section.

Gilman Cut-and-Fill System

When the large bodies of primary sulphides of great vertical as well as horizontal extent, from which about one-half of the total tonnage of the Calumet and Arizona mines is now being derived, were first opened up, they were mined by ordinary shrinkage stopes of great size, the dimensions being determined solely by the limits of the orebody. When the stope was finished the shrinkage ore was drawn out and waste filling introduced. While confined to virgin ground this method was quite safe and satisfactory, as the country rock was strong massive limestone and the walls sound. As more of the country was opened, however, these large open holes became dangerous and the method evolved itself into a combination of shrinkage and cut and fill with virtually no limit to the width of the stope or the amount of open ground above the gob. But increasing disturbance of the country rock as mining proceeded recently made it necessary to develop some safer method of mining these great sulphide lenses without too great a sacrifice of the very low costs and flexibility of the systems previously used. Consequently a cut-and-fill method was devised by Oscar Gilman, foreman of one division of the Calumet and Arizona mines, which has remedied the existing evils of the older systems and still permits of a minimum of timber, low costs, great flexibility and large daily tonnage. The method may be used wherever the ore and country rock are strong and solid and, while the width of each stope section taken is limited to 40 ft., the length and height of stopes are

limited only by the extent of the commercial ore, although for convenience sections are usually included between levels 100 ft. apart vertically.

Crosscuts are driven at 40-ft. intervals through the orebody from a strike drift, preferably in country rock in one wall of the orebody, at right angles to its general strike. A cutting-out stope is started, as shown in Section *BB*, Fig. 4, by enlarging the crosscut with stoping drills. When it has reached a height of about 8 or 9 ft. above the rail, hitches are cut in the walls and temporary so-called "Cousin Jack" stringers, *gg*, about 10 ft. long are thrown across the crosscut at 5-ft. intervals 5 ft. above the

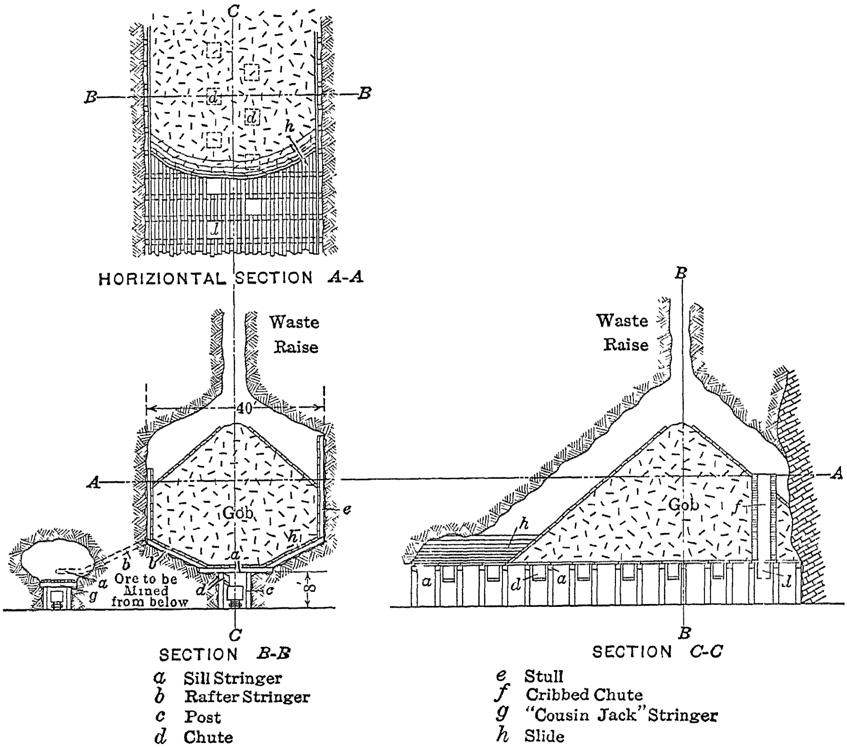


FIG. 4.—GILMAN CUT-AND-FILL SYSTEM.

rail; 4 by 10-in. timbers are used at each side of the crosscut to support these stringers where the ground requires them. Flooring is laid upon the stringers in a direction parallel to the crosscut so that ore subsequently broken may be loaded directly into cars run into the crosscut through the opening made by removing one of the center planks. This device is used only during the early stages of the operation to eliminate mucking from the floor of the crosscut before the stope has reached proportions large enough to permit of the erection of permanent sill timbering and chutes. The cutting-out stope is then enlarged by drilling deep holes

with stoping drills, maintaining an arched back and a slope of about 20° on the two sides of the crosscut to a width of 40 ft. Hitches are cut in the sides of the stope to receive 18-ft. by 12-in. by 12-in. stringers, *aa*, which are laid at 5-ft. intervals for the length of the section 8 ft. in the clear above the rail. The "Cousin Jack" stringers are then removed and if the ground requires it the 18-ft. stringers are supported by 10 by 10-in. posts, *cc*, 5 ft. in the clear. This leaves ample room for the 42-cu. ft. motor cars into which the ore is loaded in these crosscuts. As a rule, no braces are necessary between stringers, but where required 4 by 10-in. timbers are used. Then 10-in. by 10-in. rafter stringers, 14-ft. 2 in. long, *bb*, beveled as shown in Section *BB*, Fig. 4, are laid on the sloping sides of the stope with one end resting on each 18-ft. stringer; 4-in. flooring is laid over the whole and chutes, *dd*, built on both sides of the drift in each alternate set. A 10 by 10-in. brace is used between the stringers at the side of the crosscut where the chute is to be built.

While this preliminary preparation has been going on, at about the center of each section a raise has been started which is ultimately connected with the level above to be used for timber and waste filling. If waste is encountered in driving this raise it is discontinued until filling is needed in the stope. To provide for absolute safety no more than 10 ft. of open ground is left at any one time between the floor and the back of the stope. The initial enlargement of the stope takes place, as shown in Sections *BB* and *CC*, Fig. 4, beneath the waste raise, and when the 10-ft. limit has been reached, waste is admitted and forms a cone extending out toward the edges of the stope. Temporary flooring is laid upon this gob at its normal angle of repose leaving only 2 or 3 ft. between the floor and the back of the stope. The back may be supported in any weak spots while the next slice is being taken by temporary stulls or bulkheads resting upon the waste. As the chutes under this gob pile are covered over they are abandoned and the ore as broken runs upon the sloping floor and is drawn from the chutes still open at either end of the waste.

As mining exposes the sides of the stope, 8 by 10-in. stulls, *ee*, from 8 to 16 ft. in length are placed vertically at 5-ft. intervals, resting on the rafter stringers. Where one stull is placed upon another as the stope progresses upward they are securely lashed together, on four sides if possible, and each tied into the gob with timber "dummies." It is advisable to use as long stulls as practicable for every joint is a point of weakness when the next section is taken; 2-in. gob lagging is spiked horizontally to these vertical stulls before filling is admitted. In the actual process of mining it is often found that a few long holes judiciously placed toward the sides of the stope will when blasted break the arch and break a much greater tonnage than would normally be expected. When the limits of the orebody approach the vertical it becomes neces-

sary to carry up a chute or two, *ff*, next the waste wall with 4 by 10-in. cribbing through the waste filling, to be used as ore chutes. The length of the stope often becomes so great as mining progresses upward that one raise will not serve for filling the whole stope. In this case another raise may be driven through to the level above, or the outlying portions of the stope filled by driving small inclined raises off into the waste walls. It is apparent, however, that under normal conditions waste raises may be a long distance apart.

Comparative Stoping Costs[1]

Method and Conditions	Labor	Explosives	Timber	Candles or Carbide	Air	Other Supplies	Total
Square-set, oxide ore,[2] heavy ground	$0.73	$0.06	$0.34	$0.01	$0.12	$0.04	$1.30
Square-set, oxide ore, average ground..	0.60	0.05	0.25	0.01	0.10	0.04	1.05
Square-set, oxide ore, robbing timbers......................	0.61	0.05	0.16	0.01	0.10	0.04	0 97
Top-slice caving, old method, oxide ore, heavy ground..............	0 63	0.07	0 20	0 01	0.10	0 03	1 04
Mitchell top-slice caving, oxide ore, heavy ground..................	0.54	0.04	0.21	0 01	0.10	0.03	0.93
Mitchell slicing, oxide ore, average ground......................	0.51	0 05	0.20	0 01	0.06	0.02	0.85
Square-set, sulphide ore,[3] average ground......................	0 49	0.04	0.19	0.01	0.04	0.03	0.80
Mitchell slicing, sulphide ore, average ground...................	0.43	0.03	0.15	0.01	0.03	0.01	0.66
Cut-and-fill, Gilman method, sulphide ore, good ground.........	0.34	0.04	0.07	0.01	0.04	0.01	0.51
Cut-and-fill, old method, sulphide ore, good ground..............	0 32	0.04	0.04	0.01	0.04	0.01	0.46

[1] Figured on the basis of the wet tons mined. These costs obtain under normal conditions with a base wage of $4 per day for miners and $3.75 for muckers.

[2] Oxide ore in place will average from 12 to 16 cu. ft. to the ton.

[3] Sulphide ore will average from 9 to 10 cu. ft. to the ton.

If there is a great demand for ore, stoping may be commenced in the adjoining section while still mining in the original section. If this is contemplated it is better practice to drive the original waste raise nearer one end of the section so that there may be a considerable thickness of gob at that end while stoping is still in progress in the lower portion of the other end. Stoping in the new section is carried on in identically the same manner as before. The new waste raise is driven near the same end of the section as in the adjoining partially filled stope so that mining in the new section proceeds most rapidly adjacent to that part of the original stope in which the gob is thickest and most firmly consolidated. In mining alongside of this old filling care must be taken not to expose too

much of it at one time, and the stull joints must be watched. But if proper precautions are taken no difficulties are experienced in mining two or more parallel stopes simultaneously. It is inadvisable to leave a section with the idea of mining it after the stopes on either side have been finished and filled. Carrying a stope by this method between two gobbed sections is at best an uncertain operation and the risk may be entirely eliminated by taking the sections in succession as described. The ridge of ore 35 ft. wide and 14 ft. high between each stope on the sill floor is left to be mined from below. While as yet no stopes have been carried up to this point, no difficulty is anticipated as the two rafter stringers braced against each other will be virtually self-supporting and in any event it will prove a simple matter to catch them with stulls from below.

Phenomenally low costs for the Warren District have been obtained with this system of stoping. It will average about $0.50 per ton and a much lower figure has been reached under especially favorable conditions. Although slightly higher than the cost of the old shrinkage method the added consideration of safety more than counterbalances the small increase over the more hazardous system. The stope ventilation is good and the efficiency of the men high. An average yield of about 12 tons per man per shift can be maintained under normal working conditions. The daily tonnage output of each section will average 100 tons and may be crowded to 150 or even 200 tons per day.

Summary

In comparing the various methods described, the considerations enumerated in the early part of the paper must be applied to each system. The cut-and-fill system is easily the cheapest and most satisfactory in every way where conditions will permit of its use. The item of timber, usually a large one, is much reduced and the risk of fire is virtually eliminated. While any large horse of waste encountered in the ore may be kept separate and left in the gob, any attempt at close sorting is inadvisable. As in the case of the Mitchell slicing system, if the ore is fairly clean the greatest ultimate economy will be gained by mining it all as it comes without attempting to sort. The consequent low stoping cost obtained by mining large tonnages rapidly will more than counterbalance the increased cost of handling and treating a slightly lower-grade material. In neither system can the grade of ore be controlled as closely as in the square-set or top-slice caving methods. Where the ore is very irregular or badly mixed with waste, the square-set method is still found to be the most satisfactory. If the stope is in good condition a considerable proportion of the timbers may be safely extracted as it is filled.

For a heavy uniform orebody the Mitchell inclined top-slice caving system is a good one. While it still retains some of the disadvantages of

the old-fashioned top slice, it is in many ways a great improvement over the older method. Its principal advantages are the substantial saving in labor and powder, and the greater rapidity with which the ore may be mined. Its unfavorable features are those of all top-slice caving methods, the almost inevitable sacrifice of some ore, narrow work, poor ventilation in the stopes and the fact that the ground above is so badly broken as to render the expense of future work on upper levels so great as to be in many cases prohibitive. The Mitchell slicing system finds its chief application to fairly regular bodies in not too heavy ground. Under favorable conditions lower costs are obtained with this method than by any of the others in use, with the exception of the cut-and-fill method. Its saving in labor and timber is a very appreciable one over square-set stoping and the cycle of preparing, mining, filling and abandoning a section is normally a very short one. This feature, in addition to permitting a large daily output, reduces considerably the repair cost incidental to keeping a section of the country open over a long period. In short, the introduction of these methods has effected a tremendous saving to the Calumet & Arizona Mining Co. during the past few years, without the slightest sacrifice of the safety of the men or of the mines in which they have been used.

In closing the author wishes to acknowledge his indebtedness to M. W. Mitchell and Oscar Gilman, mine division foremen of this company, for invaluable assistance in the accumulation of data for this paper.

Discussion of this paper on p. 958.

Stoping Methods of Miami Copper Co.

BY DAVID B. SCOTT,* A. B., E. M., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

WHEN mining operations were first instituted in the mines of the Miami Copper Co., at Miami, Ariz., the relatively hard character of the ground in the western section of the property made it seem advisable to use a wide shrinkage stope and pillar system. The system as finally evolved was applied to the mining of 2,300,000 tons of ore by the use of stopes and pillars each 50 ft. in width. Development, mining and extraction of this reserve has been continuously in progress since 1910, and the final stages of extraction are now reached. A complete survey of all phases of this system can therefore be made. Modification in working methods was obviously necessary, and the system to be described is believed to be typical of the best features.

DEVELOPMENT

The preliminary work had three main objectives: The opening of a haulage level, the construction of a drawing-off level, and the driving of sublevels for use in stoping. In addition to these, some extra sublevel drifting was necessary to determine exactly the boundaries of the orebody, the general outline having been determined by churn drilling. This development was based on the requirements of stopes 50 ft. wide, with pillars 50 ft. wide between each stope. The length of these units varied from 200 to 500 ft.

Haulage Level

The tramming level was 50 ft. below the floor of the stopes, and the tramming lines were run parallel with the long axis of the stopes. These lines were also beneath the middle of the stopes and pillars. Each line therefore served one stope or one pillar, this arrangement making it possible to load along an entire stope without switching or uncoupling. This advantage is apparent principally during the process of stoping when rapid drawing of excess broken ore is desirable. Furthermore, the arrangement of the drawing-off level chutes at 25-ft. intervals, as shown

* Mine Efficiency Engineer.

by Figs. 1 and 2, makes this arrangement of the haulage level necessary for a minimum of development.

The drawing chutes were all on one side of the drift, each chute terminating in the drawing-off level 25 ft. above the tramming level. With

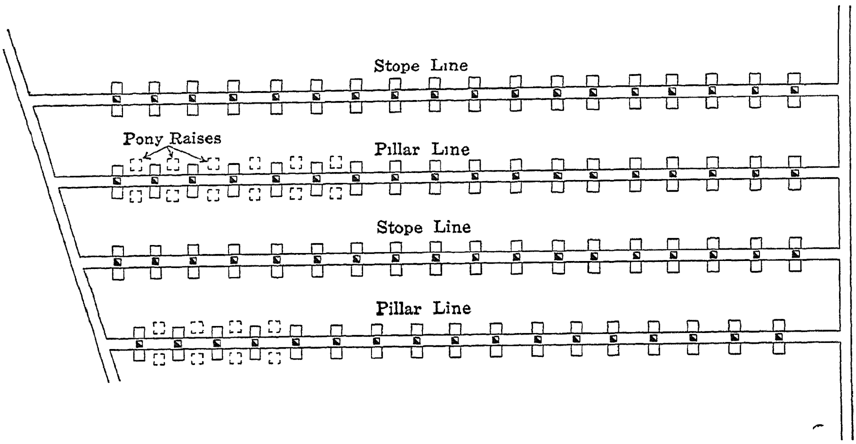


FIG. 1.—PLAN OF DRAWING-OFF LEVEL SHOWING CHUTE ARRANGEMENT. PILLAR CHUTES OF TWO TYPES.

each chute having a capacity of 15 tons, a block of seven, therefore, held a little more than the normal trainload of 100 tons. All chutes were of

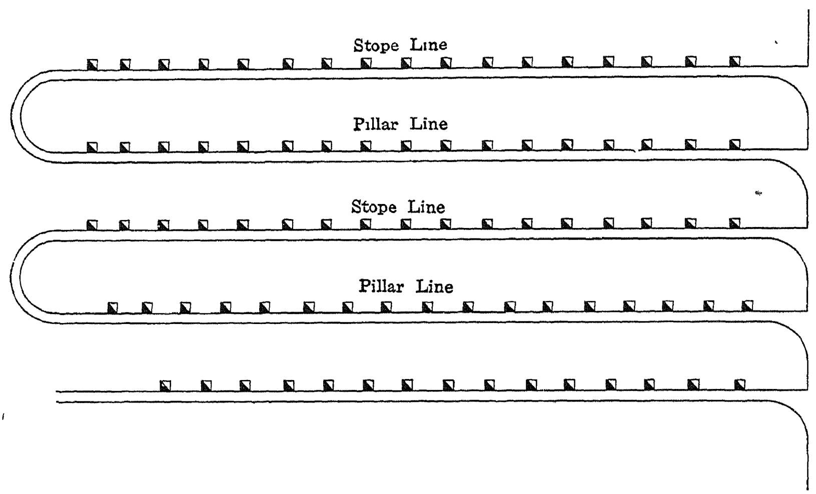


FIG. 2.—PLAN OF A SECTION OF TRAMMING LEVEL BENEATH DRAWING LEVEL AREA IN FIG. 1.

standard design, of the flat-gate or guillotine pattern with hand-operated levers. Rocks greater than 10 in. in size were seldom delivered from these chutes, and chute blasting was practically entirely eliminated on this

level. With cars of 3½ tons capacity, the loading rate was from 3 to 5 tons per minute.

In the timbering of the haulage level, 10 by 10-in. timber was used almost entirely, sets being spaced at 6-ft. 3-in. centers. The standard set which is in use at present requires a 9-ft. post and 8-ft. cap, the posts being hitched in the ground 7½ in. below the ball of rail. All drift-set posts are set at a batter of 1½ in. per foot.

Drawing-Off Level

A distinctive feature of this system was the drawing-off level, 25 ft. above the tramming level. This level was used for handling all stope and pillar ore from raises which perforated the stope floors, and delivering this ore through grizzlies to the tramming chutes. The primary drawing

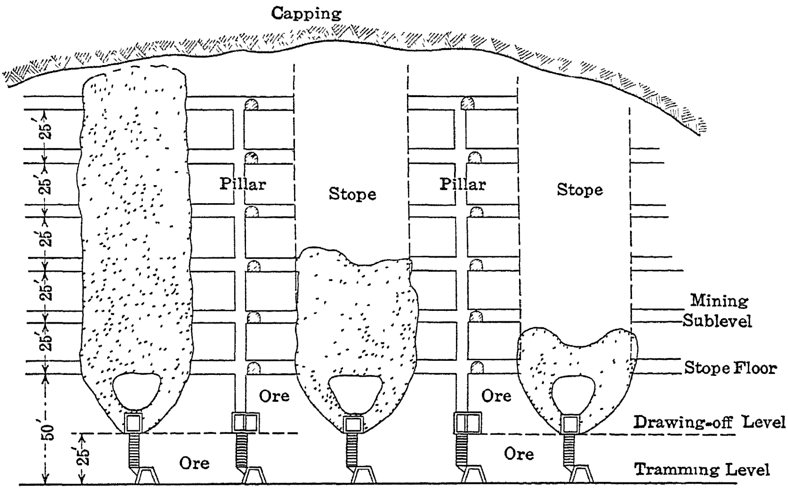


FIG. 3.—GENERAL CROSS-SECTION THROUGH STOPES AND PILLARS.

raises, as shown in Fig. 3, branch from each side of the drift and deliver directly into the chutes. The secondary or intermediate raises used extensively in the pillars were located midway between these chutes, and delivered to them by means of inclined slides.

The drawing-level drifts were driven parallel with the haulage drifts, with a fringe drift at each end to provide entry and ventilation. Stope drawing lines were located directly under the center of the stopes, so that the raises were symmetrical. Pillar drawing lines were located generally 5 ft. south of the long pillar axis for convenience in putting up vertical development raises through the pillars. This arrangement was later modified, so that symmetrical raises were also possible in the pillar lines.

Pocket Chutes.—The chutes into which the ore was drawn from the stope raises were all cribbed, 5 ft. square in the clear, 6 by 8-in. and 8 by 8-in. cribbing being used. Experience showed that the 8 by 8-in. size was the most economical over a long period. The life of this cribbing was variable on account of the quality of the ore; a representative group of chutes showed a cribbing life of 12,430 tons each.

To eliminate boulders in the tramming chutes, grizzlies were placed over the collar of these chutes on the drawing level. The grizzly opening was 18 in. square. It was found to be highly desirable to have these grizzlies placed several inches above the collar of the raise to permit easy working when ore hangs up on the grizzly. It was also found to be best

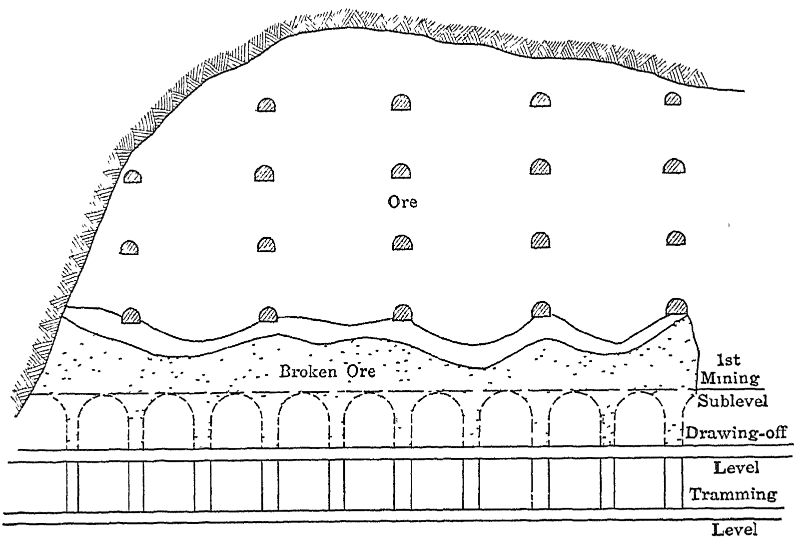


FIG. 4.—LONGITUDINAL SECTION OF STOPE DURING MINING PROCESS.

practice to have the opening in the grizzly not less than 18 in. square directly below the lip of the chute, as this is the point where "hanging up" on the grizzly starts.

The design of the drawing raises in the stopes from the drawing-off level to the stope floor differed slightly from those in the pillars.

Stope Raises.—From the floor of the drawing level, inclined raises on each side of the drift were run up to the edges of the stope, and then funnelled. Vertical center raises through the roof of the drift were originally employed, but their effectiveness is rather doubtful, and they are hoth difficult and awkward to draw. This arrangement gave sets of two raises in one plane along every 25 ft. of stope. Funnelling of these raises, as described later, to a top diameter of 20 ft. gave a practically continuous perforation of the floor of the stope. The raises were in general 6 by 6

ft. at the bottom. The back or hanging wall, constituting part of the roof pillar between the drawing level and the stope floor, had a tendency to collapse under weight, but this proved rather an advantage in providing a larger opening for the drawing of the stope ore. Timbering inside of these drawing raises for support of this brow was therefore unnecessary.

Pillar Raises.—Later practice provided raises in the pillars of the same general type as the stope raises, but the planes of these raises were not in a continuous line with the planes of the stope couples, but midway between them, as shown in Fig. 5. The earlier type of arrangement, with stope and pillar raises all in the same plane, was also extensively used and is shown in plan in Figs. 1 and 2. This arrangement is probably not as effective in drawing, however, as the other.

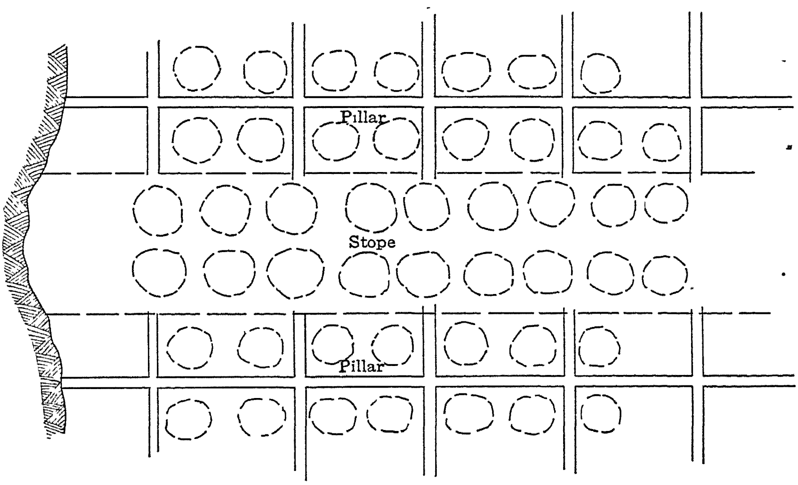


FIG. 5.—PLAN OF SHRINKAGE STOPE FLOOR SHOWING ARRANGEMENT OF RAISES.

An additional arrangement, and one largely used in the drawing of 500,000 tons of pillar ore, made use of intermediate raises. These were located 12½ ft. from the regular raises. To provide outlet for the ore, these were put up from "pony sets" in the roof of the drift, and delivered to slides which ran down into the regular 25-ft. pocket chutes. These intermediate raises provided double the ordinary number of openings into the floor of the pillar.

Timbering.—The timbering requirements of this level have shown that the drifts can be supported with 10 by 10 timbers spaced at 5-ft. centers, using 8-ft. posts and 7-ft. caps. This type of timbering remains nearly intact in some of the drawing lines after two years of service. (The chute timber details of the type which gave satisfaction, under conditions of heavy weight and wear of drawing, are shown in Fig. 11.) This set is especially effective in keeping weight from the cribbing in the pocket

chutes to the tramming level. Experience has shown that on this level a minimum amount of 2-in. lagging on drift sets should be used, and this should not be placed closer than 2 or 3 in. apart. "Pony-set" timbering for the intermediate chutes required the use of 12 by 12-in. posts in the sets. An inside height of 8 ft. in these sets was found to be more desirable both for drawing and for repair than a 6-ft. height.

Sublevels

The provision of means of entry and ventilation for the stopes governed the type of sublevel, or mining level, development. The distance between levels was fixed at 25 ft., and this distance proved to be convenient especially in the later mining of the pillars. Obviously the arrangement of drifts and crosscuts underwent some revision in opening up new ground, and as the nature of the ground became understood, the following type of development seemed to be the best for the mining of wide stopes and pillars:

Along the axis of the *pillars*, drifts were run to the ore-limits. Crosscuts at 50-ft. intervals were driven each way across the pillars to the stope boundaries. Raises for handling development ore were provided at 50-ft. intervals along the drifts. Although in earlier development crosscuts were driven entirely across the stopes, connecting all of the pillar lines, these stope crosscuts came to be regarded as unnecessary in the breaking of the stopes. If we assume therefore that no development is required in the stopes themselves, the combined tonnage of stope and pillar ore served by this development is 67 tons per foot of drift and crosscut, and 400 tons per foot of raise. Considerable secondary crosscutting and raising in addition to this, however, was necessary during the later pillar mining.

Mining

Shrinkage Stopes

The first operation was the funnelling of the drawing raises. This was done at first by cutting out a chamber above the raise and drilling 8 and 10 ft. down holes with water Leyner machines, all around the circumference of the raise. This proved unsatisfactory for the reasons that the holes did not break well, and that it was difficult to drill deep down holes of nearly vertical pitch with this type of machine. The method used almost entirely was that of setting up in the raise and drilling upholes with the same machine, in a ring around the raise. This successfully funnelled the raise to a diameter of 15 to 20 ft. Continuation of this funnelling produced a stope floor perforated with openings which practically touched on the rims across and along the whole stope. Setting

up on the broken ore, the ground was squared along the sides of the stope and any stubs between funnels blasted out. The floor of the stope was thus opened out with a back 50 ft. wide and a length of 200 to 500 ft.

Using the entries on the first mining sublevel, drilling with Leyner machines was started along each side of the stope. Steel as long as could be swung in the space was used, and holes from 10 to 15 ft. in depth were drilled upward and outward toward the center of the stope. The results of these preliminary rounds, as shown in Fig. 6, were twofold: About half of the distance to the next sublevel entries was broken along the wall of the stope, and a long belly or rib was left in the center of the stope. The principle of caving was therefore made use of to some extent in the breaking of this central rib, which was drilled in the succeeding round. After the first cut, the back was maintained on a flat incline until con-

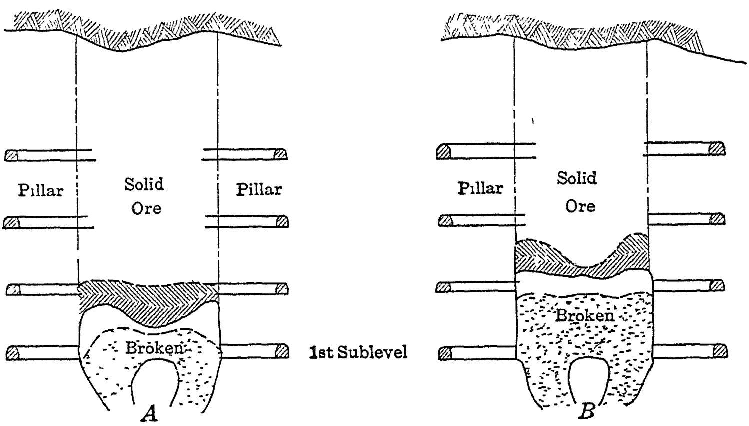


Fig. 6.—Cross-Sections Through Stope, Showing Positions of Roof.

nection was made with the pillar crosscuts on the level above. Originally some use was made of crosscuts across the stope on the level immediately above the stoping operations, in order to drill down holes into the back and thus assist the caving of the central portion of the stope. This was not satisfactory, principally because of the difficulty in getting deep holes, and in getting a wide enough reach across the back. The results did not justify the extra development expense entailed by the driving of these crosscuts.

After each drilling shift the excess of broken ore was drawn so as to allow from 6 to 8 ft. of open space between the back and the broken ore. Large blocks of caved ore were "bull-dozed" in the stope, when possible, to prevent later plugging of the drawing raises. When connection was made with each successive entry crosscut on the level above, the back and sides were squared up and the stoping continued upward in

the same manner. The stopes were carried to a maximum height of 125 ft., it being necessary in most cases to break right through to the capping. In a typical stope of hard ore, the expansion in volume due to breaking required the drawing of 39 per cent. of the ore in the stope, the latter being full to the back at completion.

Drilling was done with Leyner machines Nos. 7, 8 and 18, using steel from 10 to 16 ft. in length. The maximum footage of holes drilled per shift can be placed at 110 ft., and the average footage varied from 60 to 80 ft. according to the number of set-ups required.

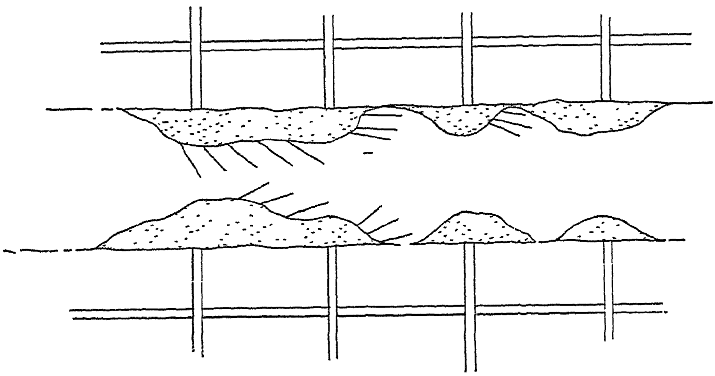


Fig. 7.—Plan of Stope During Mining Showing Drilling Methods.

Pillar Mining

The process of breaking the pillars was practically the reverse of stope breaking. Mining was started at the edge of the ore on the *top* sublevel and carried downward level by level; in each case the 25-ft. roof pillar was broken through to the broken ore above. This retreating method was desirable for two reasons: (1) It provided a safe exit from working places; and (2) it insured an even settling of the capping. As soon as the process was started, even in the hardest ground, speed of working was essential because the pillar took weight as soon as the capping became dislodged. The mining was therefore done successively on three or four levels at a time at different points of attack, the work on each level being 100 ft. nearer completion than the work on the level below it, as shown in Fig. 8.

Preliminary cross-cutting halfway between the original crosscuts was required quite generally to give better drilling faces, especially in the harder pillars. After this cross-cutting had been well advanced on the top level, drilling with water Leyners was started at the end of the pillar and across its whole width. Following the first rounds, sufficient broken ore was mucked to give entry on the sloping ore, and

machines were set up on the broken ore and long holes drilled in the back. On the top level, as a rule, the ore was not broken nearer than 10 ft. of the capping. Drilling followed strict procedure in retreating from the edges of the pillar toward the central drift, and in the retreat along the central drift as the sides were broken. The roof was carried at a slope of about 45°. All holes were drilled in the same direction, *i.e.* pointing away from the line of retreat to eliminate the chance of drilling into missed holes on the next cut. The original development raises at 50-ft. intervals were used for mucking excess ore, and generally an intermediate series of raises was put up, making the raise interval 25 ft. The final development to tonnage ratio for the completed breaking of stopes and pillars was therefore reduced by this pillar work to 45 tons per foot of drift development, and 200 tons per foot of raise development.

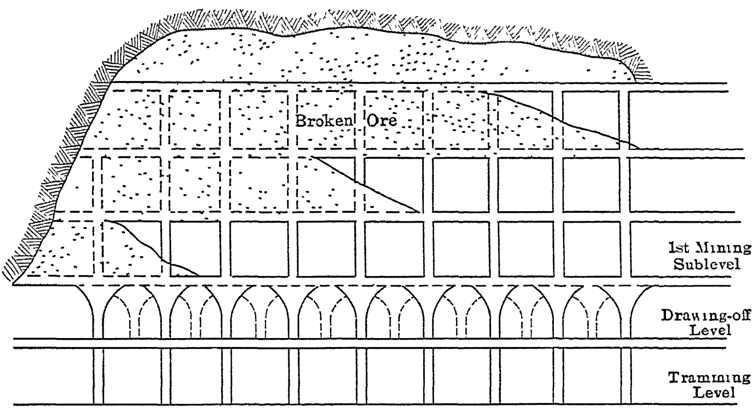


Fig. 8.—Longitudinal Section Through Pillar, Showing Mining Progress.

The first round of holes drilled from the level floor broke to about 12 ft. above the sublevel; and the second round, on which long steel could be used, increased this height above the sublevel to 20 ft. It was common practice to drill through to the broken ore above, but the remaining arch, 5 ft. or so thick, usually caved when the back was arched the whole 50 ft. of pillar width. At the start of this work the experiment was tried of drilling "down" holes in the floor of the sublevel to a vertical depth of 5 ft., to assist this subsequent caving through from the level below. It was necessary, however, to case the collars of these holes with 3-in. pipe to prevent fitchering. The breaking of these holes was unsatisfactory, so this practice of drilling floor holes was discarded.

A very important detail of pillar work was the covering over of raises as the work retreated past them. Failure to do this properly would defeat the whole system of careful breaking in allowing the capping to pipe down through the raises. Protection against this running of capping

was secured by means of stulls placed in each raise 5 ft. or so below the level on which breaking was being done, these stulls being hitched in the ground and lagged over. When the same raise was reached in the mining retreat on the level below, the process would be repeated and the stull covering above would then be blasted out, allowing the broken ore to fall to the newer and lower raise covering.

Mining on the last or sill floor of the pillars was subject to modifications imposed by the "pony-set" raises which split the distance between the regular 25-ft. raises on each side of the pillar. These raises were funnelled in retreat from the end of the pillar, along with the regular raises. This funnelling was not carried more than 25 ft. ahead of the pillar breaking, because it virtually undercut the entire width of the pillar. Machine set-ups across the raises for the purpose of drilling the back were made on 10 by 10-in. stringers which extended across the funnel. This

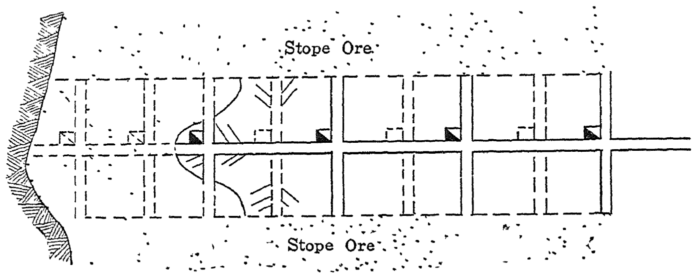


FIG. 9—PLAN OF PILLAR, SHOWING METHOD OF BREAKING IN RETREAT.

was done to insure against a sudden settling of broken ore in the raise, as occurred occasionally, with consequent danger to the machinemen.

EXTRACTION OF BROKEN ORE

Systematic drawing of the broken ore was started when about 70 per cent. of all stope and pillar mining had been completed. No drawing was permitted within 100 ft. of any mining operations. Drawing of one stope next to a pillar which was in process of being broken was tried as an experiment, and it caused crushing and swaying of the pillar even in the hardest ground.

The question of weight on the drawing-off level during the drawing period is one of the interesting features of this system. At the outset there was considerable weight in certain definite portions of the drawing-off level, and no weight whatever in other sections. This variation in different sections was due undoubtedly to a combination of the following factors: First, the height of capping above the ore in the northern section averaged 340 ft. with a thickness of broken ore of 125 ft. below it. This gave a total height of ground of 465 ft. in a mobile condition. The

southern drawing section had an average capping thickness of 250 ft. and a volume of broken ore a little over 100 ft. thick, with a total thickness of movable ground of 350 ft. The crushing in the northern section was such as to require complete re-timbering in some of the stope lines.

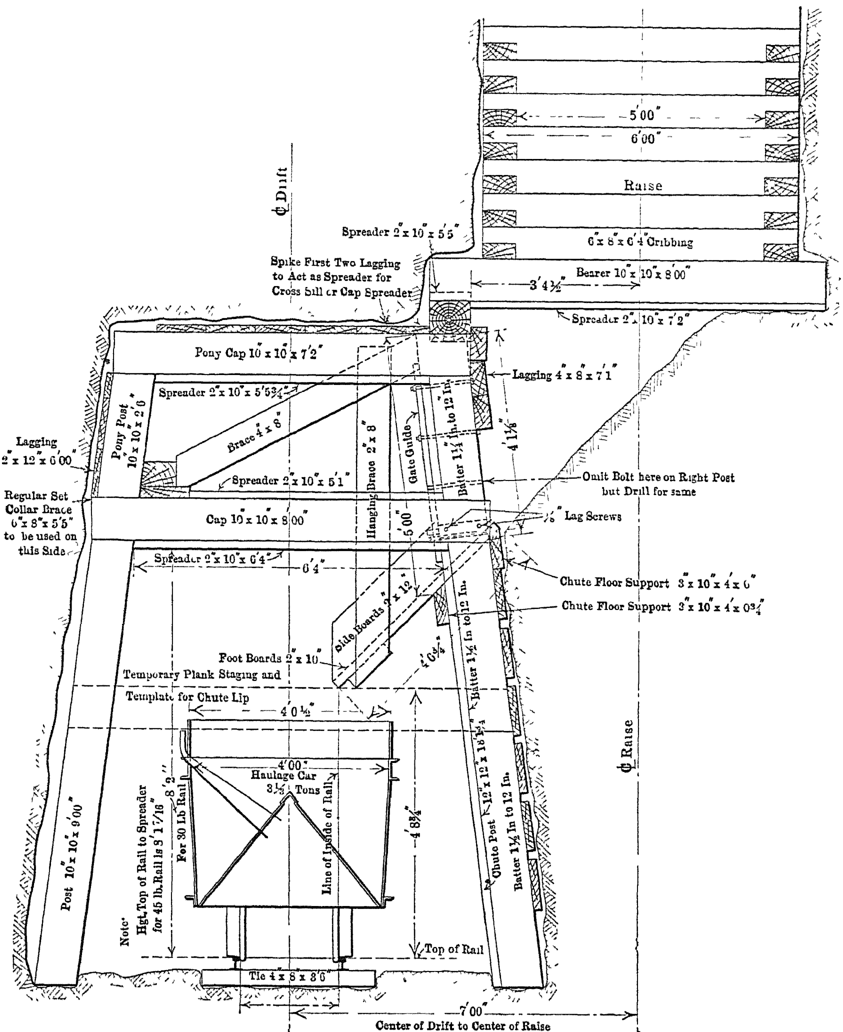


FIG. 10.—SECTION THROUGH DRIFT AND RAISE SHOWING TIMBERING OF STANDARD TRAMMING LEVEL CHUTE.

In the southern section, the weight was almost negligible; only about 20 per cent. of the lineal footage of drawing drifts in this section became heavy enough to require repair for operation. Second, in a small section where complete breaking of the pillars from top to bottom was

impossible there were some blocks of unbroken pillar above the drawing level. The sections of drawing-off drifts below these unbroken areas were always heavy, probably on account of unequal distribution of weight, or the concentration of the weight on a few points. Third, when drawing had become developed on a large scale over a considerable area, the weight did not show any increase, but on the other hand it showed a decided tendency to slack off. Upon the completion of drawing in these sections there was little evidence of weight and the drifts remained open. It is likely that the broken mass of capping and ore reached a state of equilibrium, and that the weight became very widely and evenly distributed. It is interesting to note that in one large section where all conditions during mining were satisfactory, and the pillar breaking most

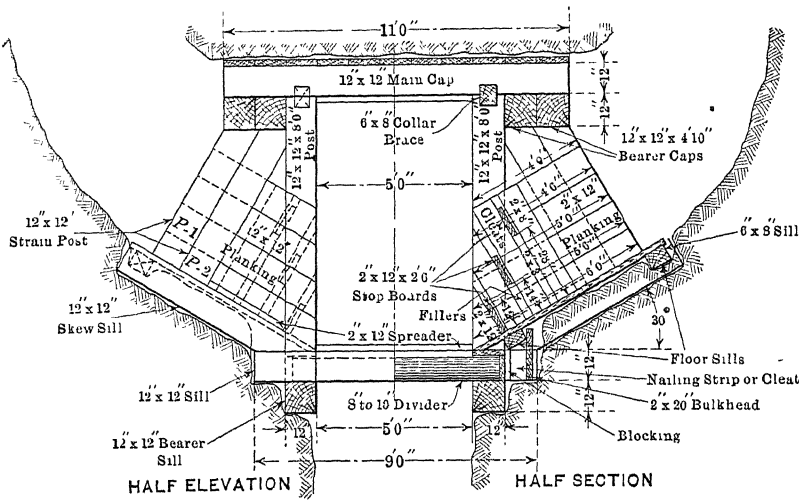


FIG. 11.—DRAWING-OFF LEVEL CHUTE SET.

complete, about 70 per cent. of the timber put in three years ago on the drawing-off level still stands.

It is apparent that weight was thrown almost entirely on the drawing drifts and not on the pillars between these drifts, as evidenced by the presence of untimbered crosscuts through these pillars. That the drawing-off level took all of the weight so that the tramming level was left intact, is shown by the present condition of the latter. Sections of the tramming level beneath badly crushed drawing-level drifts do not even require lagging in the roof, and 8 by 8-in. drift-set timbers which have been in place nearly five years show no weight. This constitutes one of the most pronounced advantages of the drawing-off level.

The repair cost on the drawing-off level did not exceed 10 c. per ton of ore drawn on any one stope or pillar line, and went considerably below

this cost on the tonnage served by the entire drawing-off level. Repair work was greatly facilitated by the presence of chutes every 25 ft. along the drifts.

Drawing Operations

Control of drawing was essential to procure an even lowering of the capping, and to permit a retreat from the western to the eastern limits of the drawing section. This required the pulling down of the capping on a more or less inclined plane which dipped possibly 15° to the western or completed portion. The drawing of a series of chutes in a plane at right angles to the direction of retreat, or across the stopes and pillars, instead

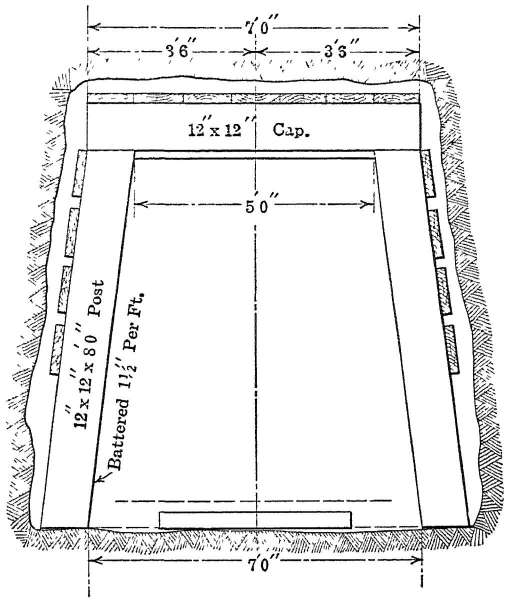


Fig. 12.—Drift Set for Drawing-off Level.

of along one whole stope, was the most satisfactory method in getting the ore to run readily.

When drawing was started there were some difficulties in getting the ore to run steadily. The nature of the ore and the length of time which elapsed between stoping and drawing, caused the chutes in some areas to pack very tightly. Some idea of the compact nature of broken ore under pressure can be gained from the fact that occasional crosscuts were driven from pillars into shrinkage stope ore, and that no timbering was required for some of these drifts in broken ore. Starting of the chutes in these stopes required some special methods. A method which resulted satisfactorily was the use of small "pony set" raises put up in the back of

the drawing-off level drift midway between the drawing raises. This involved running an inclined raise for 15 ft. on each side of the drift up to the broken stope ore. When a hole was drilled through into broken ground, the raise was funnelled immediately, and blasting of this funnelling round loosened the stope ore and usually the hung-up ore in the adjacent drawing raises. A series of these "pony set" raises sufficed to make the ore run readily in the regular raises. This method had the disadvantage that the "pony sets" took weight and were expensive. The second method, which involved no extra raises and which proved the most satisfactory, employed a system of breaking the arch of hung-up

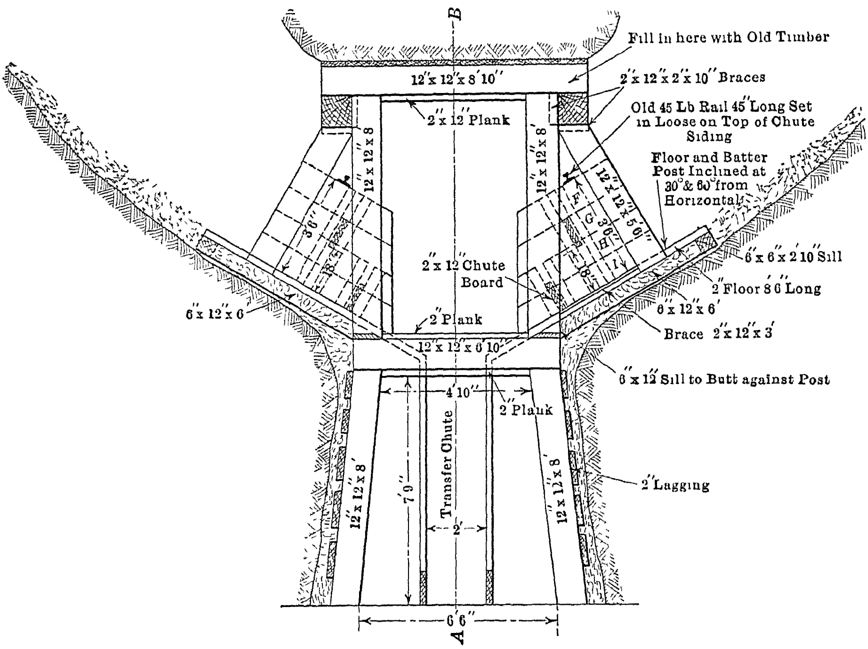


FIG. 13.—ELEVATION OF PILLAR PONY SET, DRAWING-OFF LEVEL.

ore. Entry to the back of packed ore was made through the raises, and heavy stulling was done against the hung-up ore. Small drifts were then started in the foot wall of the raise and run parallel with the long axis of the stope and at a height of about 15 feet above the drawing level. These drifts were driven in solid pillar ground and connected up a series of hung-up raises. Starting at the end of the drift the ground was drilled vertically and horizontally across the stope. When these holes were blasted, the legs between the regular raises were knocked out and an arch of broken ore left. Arches 40 ft. across and 30 to 50 ft. wide were not uncommon in this work and these arches invariably collapsed, bringing down the stope ore. In this work the hung-up ore would not cave, even if the next

raise 25 ft. away was running regularly, so that there was little danger in this work underneath the broken ore, and no accident ever occurred.

Choking of chutes was caused occasionally by packing of ore in the bottom of the inclines. It was also necessary to blast out part of the brow of the raise over the chute to prevent plugging with boulders. It was desirable to eliminate blasting by chutemen, and a separate squad of blasters, who handled all of the explosives required in chute drawing, was part of the organization.

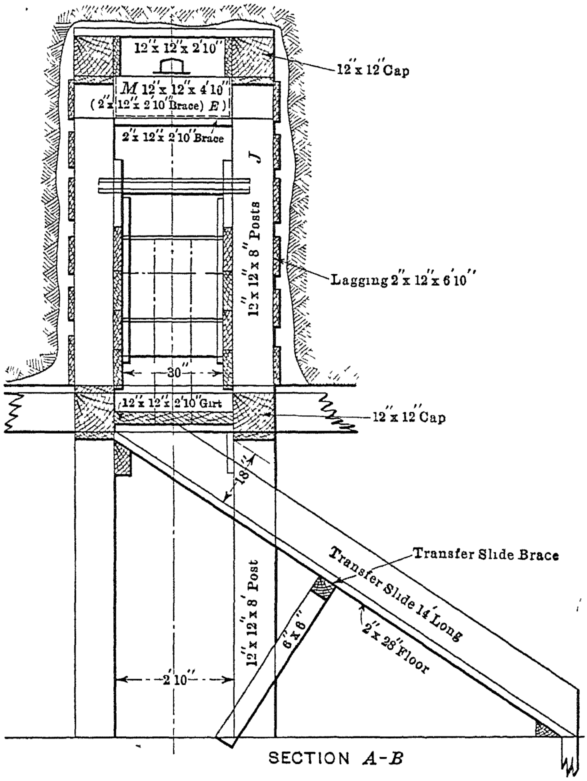


FIG. 14.—SECTION ON *A–B* FIG. 13.

Efficiency and Cost of Chute Drawing

Over a period of one year with a drawn tonnage of slightly over 400,000 tons, the average rate of drawing per shift was 118 tons per man. The monthly range ran from 94 to 142 tons per man. These figures are based upon all labor directly employed in the drawing, including chute checkers. Under very favorable conditions the tonnage per man has exceeded 200 tons for several shifts. The largest individual output ever reached on one shift was 440 tons by one chuteman.

The cost of drawing this ore into the tramming chutes amounted to 3.7c. per ton over a year's period, for direct expense of labor and explosives. Where special methods were required in the case of hung-up ore, the cost for short periods went as high as 15c. per ton.

Factors Bearing on Extraction

Figures on extraction, which involve a computed tonnage on one end and a known tonnage on the other, must be approached with some caution, yet the results of drawing, on a total output of 1,700,000 tons in one section of the work, may be of interest. This area showed an estimated recovery of approximately 95 per cent. of the original tonnage. Individual data were kept on each stope and pillar, and although the drawing from these is undoubtedly interchanged, the figures favored the stopes over the pillars by about 10 per cent.

The harder ore showed greater ease in drawing and better extraction than the softer areas, due to the greater amount of packing under pressure in the latter case. In any case the compression of broken ore is an important factor, as it increases the tendency of the drawing chutes to ravel vertically toward the capping without drawing laterally. The broken ore even under moderate pressure tended to stand nearly vertically around the rim of a drawing chute. The angle of repose, if it can be called an angle of repose in this condition, will be found to be between 80 and 90° in this class of ore. In a small number of chutes where the ore was greatly packed, the ore drawn corresponded in volume to a cylinder with a diameter of 14 ft., and a height equal to the height of ore to the capping.

It is apparent from this that a large number of drawing raises per unit of area, rather than a large area in the funnelled mouth of the raise itself, is desirable for the best extraction.

Secondary Recovery of Ore

It became obvious at the conclusion of drawing that a certain amount of clean broken ore remained in the stopes and pillars. This remainder probably stood in the shape of a wedge directly over the drawing drifts, and also along the boundaries of the old stopes and pillars. To recover the ore above the drawing drifts, the backs of the latter were shot down through to the floor of the stope. This work was done in retreat, and the ore mucked and drawn on the incline into the tramming chutes. This work cost from 15 to 25 c. per ton, and the output per man averaged in general 40 tons. Results obtained from these operations confirmed the view that the clean ore stood in a series of wedges or cones, which had an apparent base of 20 ft. and a height of 50 ft.

To recover the ore standing where the old stopes and pillars joined, a series of intermediate drifts midway between the original drawing lines was driven. These drifts were at the same elevation as the drawing-off level. Dumping chutes and supply raises were provided from crosscuts on the tramming level at 200-ft. intervals. The drifts were supported with 12 by 12-in. timbers set at a distance of 4 ft. 2 in. center to center, and chutes were placed in each set staggered along the drift. The back was broken on top of the lagging, and a small shrinkage stope about 20 ft. wide carried up until loose ground was encountered, or until caving occurred. The ore drawn from the chutes was trammed in 1½-ton cars to the main tramming raises, the maximum tram being 100 ft.

This work, which is now in operation, marks the completing stage of mining hard ore by the wide stope and pillar system.

Coöperative Effort in Mining

BY JOSEPH P. HODGSON, BISBEE, ARIZ.

(Arizona Meeting, September, 1916)

Introduction

SINCE about 70 per cent. of the total cost of mining is due to underground work which is out of sight, it is essential that expenditures should be made here to the best advantage. A great many mistakes have been made which were expensive, but perhaps these have not even been detected by mine owners. In the future these may be minimized to some extent, especially if there is coöperative effort on the part of each employee.

I have been interested in this subject for some years and have tried to deduce principles and methods. These have proved highly advantageous and may also be of help to others. Some corporations, like the United States Steel, and the Cleveland-Cliffs Iron Co., have done a great deal in this field by having stated meetings of their organizations, with the view to interchanging ideas among their men and educating them along the lines of concerted effort.

Since no man knows all about the various branches of mining, it is advisable that frequent meetings of the operating department be held for the interchange of ideas and that all members of the organization have an opportunity to express themselves on the various subjects. It has often been found that in discussions between heads of departments, foremen and bosses, many valuable suggestions may be obtained which promote the best interests of the company. The method regarded as the most conducive to coöperative effort, and pursued by the organization, is to analyze and develop the different phases of various mining subjects at regular meetings.

Mine Foremen's Dinners

In the latter half of 1912, the Copper Queen Consolidated Mining Co. gave monthly dinners to the men in the operating departments. Those who attended were superintendents, foremen, shift bosses, engineers, geologists, and heads of all departments, making a total of about 75 men. After the dinner, lectures were given on subjects which pre-

sented the most difficult problems. The purpose of the meetings was to get the men of the different departments acquainted, talk over the various problems, and standardize methods as much as possible.

Lectures

At first the meetings were formal and those who attended them had little experience in the way of presenting and advancing ideas, but after a few meetings the men became better acquainted with each other and a systematic plan for speaking on various subjects was mapped out. It soon became commonplace for the men to express their opinions freely. Since the grade of the ore was one of the most perplexing problems, Mr. Rutherford, the Smelter Superintendent, delivered the first lecture on the cost of smelting ore and waste. Later, Dr. Douglas spoke of the early history of the Copper Queen, and Walter Douglas spoke of some changes which led to profit-sharing. A Sunday excursion to Douglas was arranged and the importance of the grade of the ore was further explained on the ground, at the smelter.

Papers and Discussions

The following subjects outline the papers and discussions that were delivered from the time that the monthly dinners were started, up to the present:

Compressed Air; Mine Fires; Ventilation in Metal Mines; Mining Methods, Progress in Efficiency; Safety First, Costs, etc., in Lake Superior Iron and Copper Mines; Costs of Mine Supplies; Lagging in Drifts and Stopes; Method of Covering Timber Compartments for Maximum Ventilating Purposes; Candles and Carbide Lights, Efficiency and Costs; Standard Raises, Manways, and Timber Compartments; Standard Sizes of Timber; Electrical Costs Underground; Standardizing Steel and Machines; Geology of Bisbee Ore Deposits, Field Excursion; Stoping Methods at Morenci, Miami, Ray, and Inspiration.

Copper Queen Mining Conference

On July 1, 1915, the Copper Queen Mining Conference was formed for the purpose of instructing shift bosses, engineers, and others, in the essentials of mining. This department is an expansion of the mine foremen's dinners. At the dinners the papers and discussions are brief and of a general nature, and half the evening is devoted to entertainment, while at the mining conference papers are discussed in greater detail, and problems receive further study. Attendance at the conference is optional; however, it is evident that all who attend receive instruction

beneficial to themselves as well as the company. Instruction is imparted in the form of lectures, reviews, discussions, readings, results of investigation in mining methods, etc. Lectures are given every Tuesday evening and the meeting lasts one hour. As the shift changes every two weeks, lectures are repeated for the benefit of those who are on the night shift. Those who attend are the same representatives as mentioned at the mine foremen's dinners. About 75 per cent. of the men in the operating department attend the mining conference. The following subjects were taken up during 1915:

Geology; Classification of Ground; Types of Machine Rounds; Time Study in Drilling; Bonus System; Caps and Fuse; Powder; Drilling Machines; Steel; Milling.

The present outline of subjects for discussion during the next five months (Jan. 1 to May 31, 1916) is as follows:

Mine Ventilation; Sampling; Safety First; Mine Timber; Water Lines; Costs of Mine Supplies; Leaching; Electricity; Smelting; Management.

Results

All the men in an official capacity have become well acquainted with each other. The monthly as well as the weekly meetings usually bring forth fresh ideas, many of which are carried out from time to time. For instance, the best methods of putting in manways were suggested and worked out by bosses; also methods of repair work were first suggested by bosses and foremen. The central blacksmith's shop is the product of many suggestions by a number of those in the operating department. The same is true of the central sawmill, the ventilation system, the daily stoping costs, etc. Whatever has been accomplished in the way of reduction of costs, or the working out of new things, we believe has been due, in a large measure, to the efforts of the members of the various departments in planning and working in harmony, which is really the result of coöperative effort.

Discussion

Joseph P. Hodgson.—As to "Safety First," I did not deal with that, but I will say that in 1913, the number of men killed was nine, and that year the "safety first" program was commenced; the next year, 1914, the number was reduced to five. In 1915, I am pleased to say that the number was reduced to two in the Copper Queen's entire property, and we have none so far this year.

The Chairman (G. F. G. Sherman, Bisbee, Ariz.).—I can say, further, that we have from eight to nineteen men per month incapacitated for more than 14 days. Those are classified in this State as serious

accidents. The number of men injured has apparently increased, but that is due to our improved method of recording and classifying accidents, no matter how slight.

C. W. Goodale, Butte, Mont.—The importance of coöperation among the members of the staff of an industrial organization is well brought out by Mr. Hodgson's paper.

The Anaconda Copper Mining Co. has done something along this line, but has not gone as far as the Copper Queen Co. At the Anaconda works, on the last Tuesday of every month, the manager meets the heads and foremen of all departments, for the purpose of discussing costs and efficiency. The superintendents are asked to give the reasons for any changes in costs, and also to suggest methods by which greater efficiency may be obtained. If any member of the staff has been away visiting other plants, he gives an outline of his observations, and then a general discussion follows.

In the efforts of the company to improve working conditions and effect a reduction in the number of accidents, coöperation is of the greatest importance—not only among the foremen and shift bosses, but among the miners and smeltermen.

When the Bureau of Safety was established, about 3 years ago, our first step was to make an analysis of causes of accidents, with a view to attacking the principal causes with special vigor. A review of accidents in the Butte mines for the preceding year showed the following figures:

	Per Cent.
Fall of ground	43.21
Caught by mine cars	12.97
Handling tools	14.54
Handling material	11.87
Falling	10.00
Falling timber	3.62
Machinery	1.11
Around cages	0.74
Explosives	0.70
Miscellaneous	1.24
	100.00

In the meetings of the safety engineers and foremen these figures were considered, and the fight against accidents began. It was soon found that in order to make the miners coöperate in this movement, we should have to inflict some penalties on them for failure to look out for their own safety. For instance, when men are found working under conditions which are unsafe, and which they can remedy, they are laid off 7 days, and for the second similar offense the penalty is a layoff for 14 days. For the third offense they are discharged.

We have effected quite a satisfactory reduction in accidents in the mines of the company. During the year 1915, we had 1.42 fatal and serious accidents per 10,000 shifts, and 1.09 in the first half of 1916—a reduction of 23 per cent.—and this in the face of a mine fire which occurred in February, 1916, and which caused the loss of 21 lives.

In the reduction works our efforts have not accomplished so much. Perhaps one explanation is that the causes of accidents in reduction works are more varied in character than in the mines, and it has been difficult to attack any one particular cause with special emphasis. Furthermore, it should be stated that a large amount of construction work has been going on during the last 3 years, and I think it is generally admitted that there are more risks in such work than in operation—particularly when construction and operation are going on together, as has been the case in some of our departments.

At the Anaconda works the company has spent more than $60,000 in safeguards which have been suggested at meetings of the Safety Committees, and the effect will doubtless be shown in accident records as time goes on.

Charles A. Mitke, Bisbee, Ariz.—The practical man's lack of opportunity presents a difficulty to his satisfying his interests in technical subjects. The object to be attained in introducing this course was to show that there is no dividing line between the so-called practical and technical sides of mining; that if any ideas, devices, methods or applications are practical, then they are also technical because all practical methods have a technical basis. The converse is also true that all technical methods which have commercial value and are worth introducing are practical and have a practical basis. However, the terms practical man and technical man still exist, and it is only by coöperation of both classes that the best results can be obtained through their combined coöperative effort. Four years ago none of our shift bosses or foremen would attempt to write or read a paper at any meeting. However, after certain members of the operating staff presented papers, they were also induced to prepare papers. These men were given all possible assistance. Their subjects were first discussed with them individually so as to bring out the important points and then were written and revised to emphasize these points and in their final form presented at the meetings. Two courses were pursued during the past year—a general course at the mine foremen's dinners and a definite course in mining methods, etc., at the Copper Queen mining conference. An example of subjects taken up in the general course was a lecture on "The Importance of Keeping up the Grade of the Ore," while an example of the subjects used at the conference was the discussion of "The Details of Doing Development Work." At mining conferences a constant effort was made

to bring out the basic principles which govern the varied mining operations here as well as in other fields. After they were taken up, the application of these principles was then considered in connection with specific and local mining problems. It was reasonable to believe that this plan would be interesting and at the same time would present the subject from a broader point of view. When members of the conference were asked to prepare papers, they received every encouragement to take pains and make them as thorough and complete as possible. It also led to another field. This led in some cases to suggestions from the shift bosses as to the details of our operations. These suggestions from the shift bosses were then embodied in papers and presented at subsequent meetings with the guidance and assistance of the instructor. During this coming year we plan to increase the scope of the field. A course to miners will take up the principles and first course in mining methods; a course for shift bosses and engineers will give more advanced work in mining problems and greater detail in mining methods; and a course for the foremen's dinners will include subjects of a broader nature that will apply to all concerned. The course at the foremen's dinners consists of lectures, while that at the Copper Queen mining conference for shift bosses and engineers is a combined course of lectures, problems, readings, study, etc. There are many things that a shift boss ought to know and, since they can only be learned through study and application, these courses have been instituted.

C. E. Arnold, Miami, Ariz.—Mr. Chairman, Mr. Hodgson made a practice of putting on some instructors. It would be interesting to hear what the results of that have been.

J. P. Hodgson.—Some months ago we took up the study of classifying ground. After the ground was classified we experimented in the different classes of ground in an attempt to standardize our drilling methods underground. It was found that in trying to standardize we bumped into old miners who, in their opinion, knew more about it than we did. So we decided to take the line of least resistance. When we found a man unwilling to learn a new method we did not fight him. We put him in a stope to work and took young men and trained them. We appointed several instructors and have them in the mine. They take young men who want to get ahead and instruct them for a period of 10 days to 2 weeks, then we supply them with a machine and equipment and put them in a drift, or raise, as the case may be. They get miners' pay and a bonus, depending upon the classification of ground upon which they are working and advance made. We have found this to be beneficial, and we have decided in the future rather than to fight our miners, to help them.

D. W. Brunton, Denver, Colo.—I would like to give some figures on speeds maintained in tunneling operations where persistent effort in teaching the miners who were engaged in driving tunnel headings resulted in an increase of speed from 250 ft. per month to 750 ft. per month, and in ordinary ground, after the workmen were thoroughly trained, there was little or no difficulty in maintaining speeds of from 600 to 750 ft. per month, with three 8-hr. shifts—in headings 7 ft. high by 8 ft. wide.

The Chairman.—This subject verges on that allied subject, efficiency engineering, with which we have been experimenting. A characteristic of that subject is that it does not take very long to find out how little you know about a great many standard operations, and that they are not done as they should be done, and are not yet as we would like to have them. I don't know whether our experience has been like that of others or not.

Diesel Engines versus Steam Turbines for Mine Power Plants

BY HERBERT HAAS,* SAN FRANCISCO, CAL.

(Arizona Meeting, September, 1916)

CHEAP power is essential to large-scale mining and metallurgical operations, particularly where fine grinding of large tonnages has to be resorted to, as is the case with an increasing number of mines treating disseminated copper ores, or low-grade gold ores, in the grinding of which considerable power is consumed. A comparison of the cost of generating power with Diesel engines and steam turbines and a brief discussion of the different factors governing the profitable use of either type of prime mover may be timely in view of the importance of a cheap supply of power to the mining industry, and the prominence that has lately been given to Diesel engines in America.

As many of our most important mines and largest producers of metals are located in sections where neither cheap coal nor water power is available, and the price of oil fuel is increased by long hauls, the Diesel engine will become a prime mover of increasing importance. This applies particularly to the Southwestern States, where an ample supply of cold water for condensing purposes is unusual and local climatological conditions interfere in securing the high vacua essential to high turbine efficiency.

About 12 times as much water will be required for condensing purposes with steam turbines under general conditions as is needed for jacket-cooling of two-stroke cycle Diesel engines; four-stroke cycle Diesel engines require only one-twentieth of the cooling water.

Another condition favoring the use of Diesel engines is the high load factor at which most mine power plants operate. The influence of load factor on the cost of generating power is shown in Tables 1 and 2 and in Fig. 1.

The higher the cost of fuel and the load factor, the greater will be the proportion of the fuel to the other costs making up the total cost of generating power. Capital charges (interest and redemption), maintenance, operating labor, and lubrication are practically constant amounts regardless of the load-output of the station; the fuel consumption and its cost are practically the only variables with the load.

The costs here given are based on conditions as generally found in the Southwestern States, and on Diesel engine operating experience in European countries, where this type of prime mover has reached a high state of perfection, units upward of 4,000 hp. having proved entirely successful in operation.

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* Consulting Engineer.

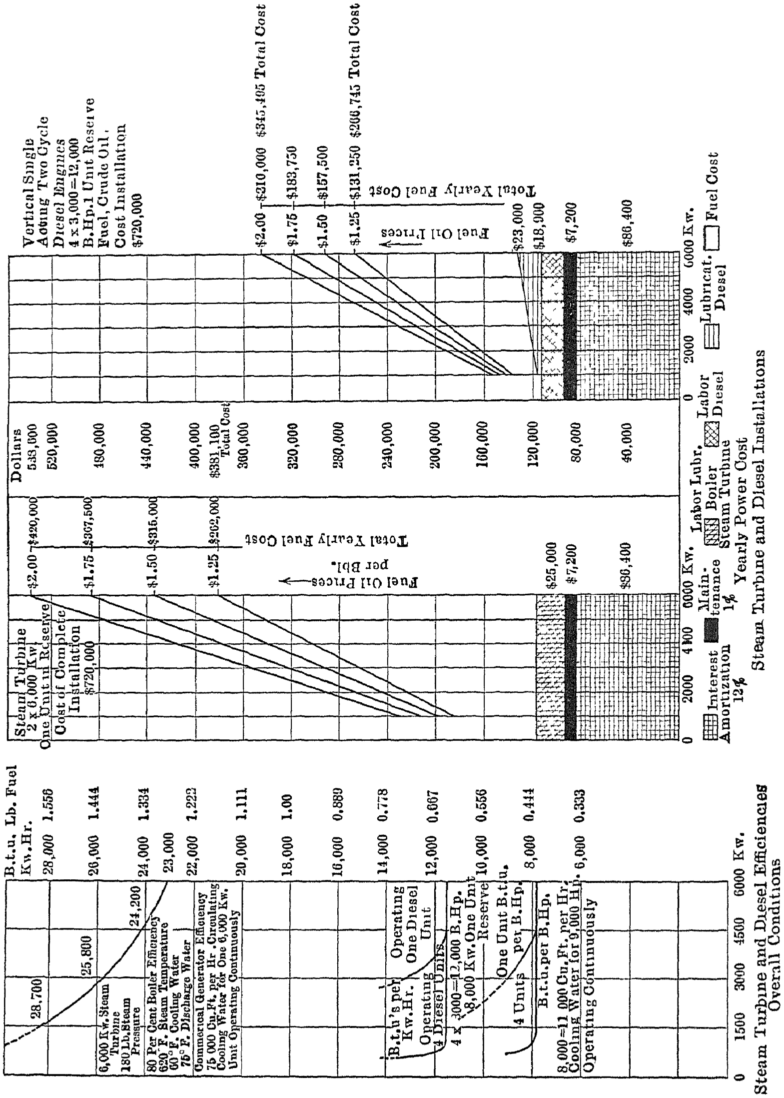

Fig. 1.—Comparison of Steam Turbine and Diesel Engine Efficiencies and Costs.

TABLE 1.—*Cost of Generating Power with Four* 3,000 *b.hp.* (2,000 *Kw.*) *Sulzer Diesel Engines Direct-Connected to Alternators and Exciters*

Costs are based on the following conditions:
Cost of complete plant $720,000
Cost of fuel, $1.25 per bbl. of 320 lb.
Cost of lubricating oil, 35 c. per gallon.

			Per Month	
Labor—	1 chief engineer	at $200	$200	
	3 watch engineers	at $150	450	
	3 switch-board men	at $125	375	
	3 helpers and oilers	at $ 90	270	
	2 machinists	at $140	280	
	12 men, total per month and year.... . . .		$1,575	$18,900

Maintenance—1 per cent. of cost of installation, per year .. 7,200
Interest, at 6 per cent. per annum 43,200
Amortization at 6 per cent. and re-invested at 4½ per cent. to redeem capital of $720,000 in 15 years 43,200

The kilowatt-year, figured at 8,760 kw.-hr. Four, three, two, or one unit operating, respectively, furnish full, ¾, ½ or ¼ load, generating 70 million, 52½ million, 35 million and 17½ million kw.-hr. per year respectively. Three units to operate continuously, one unit to serve as stand-by. Cooling water in circulation for 9,000 hp. operating continuously, 8,000 to 11,000 cu. ft. per hour without re-cooling arrangements.

	12,000 hp. or 8,000 kw		9,000 hp. or 6,000 kw		6,000 hp or 4,000 kw		3,000 hp. or 2,000 kw.	
Kw.-hr. per year .	70,000,000		52,500,000		35,000,000		17,500,000	
Barrels... . . .	140,000		105,000		70,000		35,000	
	Cost		Cost		Cost		Cost	
	Per Year	Per Kw-hr, Cents	Per Year	Per Kw.-hr, Cents	Per Year	Per Kw-hr, Cents	Per Year	Per Kw-hr Cents
Fuel cost	$175,000	0.2500	$131,250	0.2500	$ 87,500	0 2500	$ 43,750	0.2500
Labor .	18,900	0.0270	18,900	0 0360	18,900	0 0540	18,900	0.1080
Lubrication ..	30,660	0.0438	22,995	0.0438	15,330	0.0438	7,665	0.0438
Maintenance .	7,200	0.0103	7,200	0 0137	7,200	0.0206	7,200	0.0412
Total direct cost.. ..	$231,760	0.3311	$180,345	0.3435	$128,930	0.3684	$ 77,515	0.4420
Interest and amortization..	86,400	0.1234	86,400	0 1646	86,400	0.2468	86,400	0.4936
Total cost	$318,160	0.4545	$266,745	0 5081	$215,330	0.6152	$163,915	0.9356
Cost per kw-yr .	39 81	......	44.51	. .	53 89	......	83 96	.
$1 50 oil.. .	210,000	0 3000	157,500	0.3000	105,000	0.3000	52,500	0 3000
Total cost .	343,160	0.5045	292,995	0 5581	232,830	0 6652	172,665	0 9856
$1 75 oil... .	245,000	0.3500	183,750	0 3500	122,500	0.3500	61,250	0 3500
Total cost . .	388,160	0.5545	319,245	0.6081	250,330	0.7152	181,415	1.0356
$2 oil..	280,000	0.4000	210,000	0 4000	140,000	0.4000	70,000	0.4000
Total cost .	$423,160	0.6045	$345,495	0 6581	$267,830	0.7652	$190,165	1 0856

TABLE 2.—*Cost of Generating Power with Two 6,000-Kw. Steam Turbines*

Costs are based on the following conditions:

Cost of complete plant....................................	$720,000
One unit operates continuously at full load, one unit is stand-by.	
Cost of fuel oil, $1.25 per bbl. of 320 lb.	
Labor, lubrication, boiler upkeep, per year........	25,000
Maintenance, 1 per cent of cost of installation, per year	7,200
Interest and amortization, 12 per cent. of cost of installation, per year..................	86,400

Kilowatt-year figured at 8,760 kw.-hr. Turbines to operate at 180 lb steam pressure, 620° F. steam temperature, 80 per cent. boiler efficiency, 60° F. inlet temperature, 75° F. discharge temperature of condenser circulating water; quantity of cooling water per 6,000-kw. unit : 74,000 cu. ft. per hr. Commercial generator efficiencies.

	8,000 kw.		6,000 kw.		4,000 kw.		2,000 kw.	
Kw.-hr. per year..	70,000,000		52,500,000		35,000,000		17,500,000	
Barrels fuel oil .	300,000		210,000		150,000		83,700	
	Cost		Cost		Cost		Cost	
	Per Year	Per Kw -hr , Cents	Per Year	Per Kw.-hr , Cents	Per Year	Per Kw -hr., Cents	Per Year	Per Kw -hr , Cents
Fuel cost .	$375,000	0 5357	$262,000	0 5000	$187,000	0 5357	$104,625	0 6000
Labor, etc	25,000	0.0357	25,000	0.0476	25,000	0 0715	25,000	0 1430
Maintenance .	7,200	0.0103	7,200	0 0137	7,200	0 0206	7,200	0 0412
Total direct cost	$407,200	0 5817	$294,700	0.5613	$219,200	0.6278	$136,825	0 7842
Interest and amortization	86,400	0 1234	86,400	0.1646	86,400	0 2468	86,400	0 4936
Total cost	$493,600	0.7051	$381,100	0.7259	$305,600	0 8746	$223,225	1.2778
Cost per kw -yr	61 75	......	63.50	......	76 60	.. ...	111 70	. .
$1.50 oil . . .	450,000	0 6430	315,000	0.6000	225,000	0 6430	125,500	0 7171
Total cost ..	568,600	0 8123	433,600	0 8259	343,100	0 9818	244,200	1 3949
$1 75 oil .	525,000	0 7500	367,500	0.7000	262,500	0 7500	146,375	0.8342
Total cost . . .	643,600	0.9194	486,100	0.9259	380,600	1.0890	265,124	1.5120
$2 oil.... . ..	600,000	0 8571	420,000	0.8000	300,000	0 8571	167,250	0 9513
Total cost ..	718,600	1.0257	538,600	1.0259	418,100	1.1962	286,050	1.6291

The last three double columns show the effect of increased cost of fuel oil on power cost, the total fuel cost being shown, as well as the total cost of power, which is merely increased in proportion to the increased fuel expenditure

This paper will not deal with questions of design, nor types of engines, as much space would be needed to do justice to the large number of excellent makes of engines. It is assumed that the reader has a general knowledge of Diesel engines.

The reasons favoring the selection of two 6,000-kw. steam turbines and four 2,000-kw. Diesel engine generating sets, can be briefly stated as follows:

1. Uninterrupted power service is essential, as failure of power would entail far greater losses in operating revenue than the interest on the cost of the reserve turbine unit.

2. The selection of two 6,000-kw. rather than three 3,000-kw. steam turbines is made on account of the greater efficiency of a 6,000-kw. over two 3,000-kw. units, the load to be supplied being nearly 6,000-kw. continuously. (In plants with fluctuating loads which can be supplied by one or more smaller units operating in parallel during different portions of the day, each unit operating at or near its full-load capacity, a number of small units is justified; such diversified load conditions, however, apply mainly to central stations in cities, rather than mine, mill, and smelter power plants, which have invariably a high load factor during the entire day.)

3. The reserve boiler capacity would be an additional third of the boiler capacity in continuous operation, regardless of size of turbines selected. The extra cost of the surplus turbine capacity by reason of installing two 6,000-kw. instead of three 3,000-kw. turbines is therefore moderate when taking future power costs and other factors into consideration.

4. The selection of four 2,000-kw. Diesel engine units is justified on the ground that larger engines are not any more efficient in fuel economy, and the relatively much higher cost of this prime mover[1] makes it policy to reduce reserve capacity to a safe minimum. An excessive number of small Diesel units should also be avoided, as the cost per kilowatt of installed power-plant increases, and the multiplication of cylinders, valves and engine parts to be taken care of increases maintenance and labor charges.

The heat utilization and cooling-water requirements of Diesel engines and steam turbines at full load are graphically shown in Fig. 2 for units of 2,000 hp.

In general, the selection of either type of prime mover will be governed by the following economic considerations:

1. Fuel is of chief influence on the total power cost where both the B.t.u. price[2] and load factor are high. These conditions favor the use of the Diesel engine.

2. Interest and redemption (amortization) are of chief influence on the total power cost with low B.t.u. price and low load factor; this applies

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[1] A Diesel engine with its direct-connected compressor, alternator and exciter is a complete prime mover, and should be compared with a complete steam-power plant comprising boiler and condenser equipment, turbine and generator. Diesel engines are simple in that the installation is compact, comprising few parts, and occupying only about 50 to 60 per cent. of the space of a steam-turbine installation of like power.

[2] By B.t.u. price is meant the price or cost for a given number of B.t.u., say 1,000,000 B.t.u., all fuel prices being reduced to this standard of comparison.

particularly to stand-by plants, which are operated only occasionally or have to supply recurring peak loads. The installation cost of such plants must be kept as low as possible so as to avoid heavy capital charges distributable over a relatively small kilowatt-hours output of the station. Such conditions favor the steam turbine.

3. Exceptions to (2) are cases where the constant and instant readiness of the Diesel engine give it preference, and installation cost is of secondary importance. Here we would have to balance the cost of keeping boilers under steam continuously against the difference of interest charges of steam-turbine and Diesel engine plants.

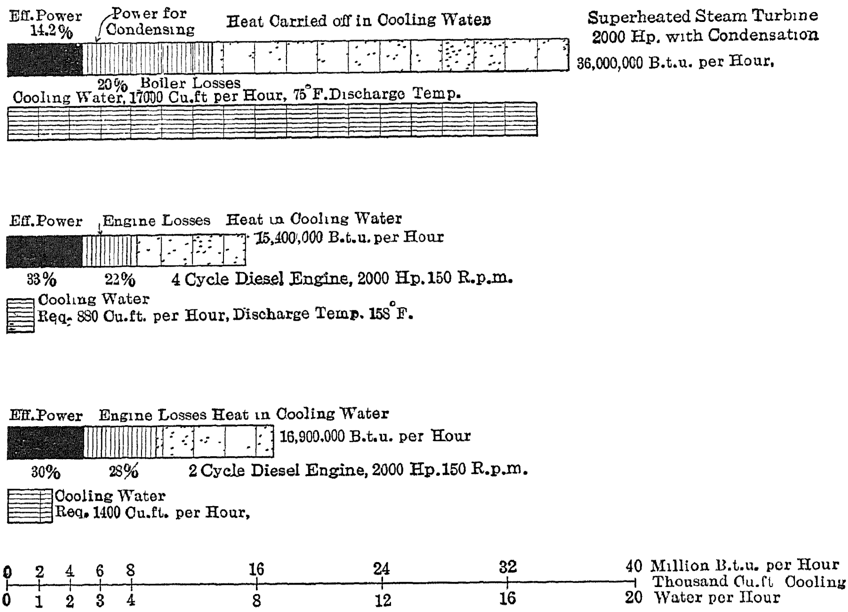


FIG. 2.—COMPARATIVE EFFICIENCIES OF STEAM TURBINES AND DIESEL ENGINES OF 2,000 EFFECTIVE HORSEPOWER.

4. Power plants located at the source of the fuel, either in the oil fields, or at the coal mine, and therefore enjoying the advantages of a very cheap fuel supply, will select prime movers costing least to install, *i.e.*, steam turbines, since fuel expenditures will weigh less than interest and redemption charges.

5. In many cases combination plants using Diesel engines for supplying the continuous and nearly constant main load, and steam turbines for furnishing periodically occurring peaks by the use of high-duty boilers with large water and steam spaces, capable of being forced when necessary, will prove most profitable. Thus, as an example, the periodical peaks produced in hoisting may be taken care of by a turbine floating on

the line and operating in parallel with Diesel engines that supply the main load and operate constantly at or near full load.

6. Steam power will remain the cheapest power wherever waste-heat gases are available, as, for instance, gases from reverberatory smelting furnaces, where nearly one-half of the fuel used in smelting can be utilized for steam generation. Nearly 3,000,000 B.t.u. for every ton of charge smelted are thus available for steam generation, or about 150 hp.-hr. per ton of charge.

7. Up to capacities of 1,000-hp. steam turbines can compete with Diesel engines only in special cases, such as supplying exhaust steam for heating purposes. For such small units, particularly for greatly varying loads, high-grade reciprocating steam engines are preferable. For larger plants, from 1,000 to 10,000 kw. capacity, careful analysis must be made of the relative advantages of Diesel engines and turbines, a knowledge of the load factor, fuel prices and water conditions being necessary. Power plants larger than 10,000 kw. using units from 6,000 kw. upward, preferably use steam turbines, unless a combination of high load factor, high fuel cost, and poor water conditions favor a Diesel plant. This is a combination of special conditions not frequently met.

Fuel Costs

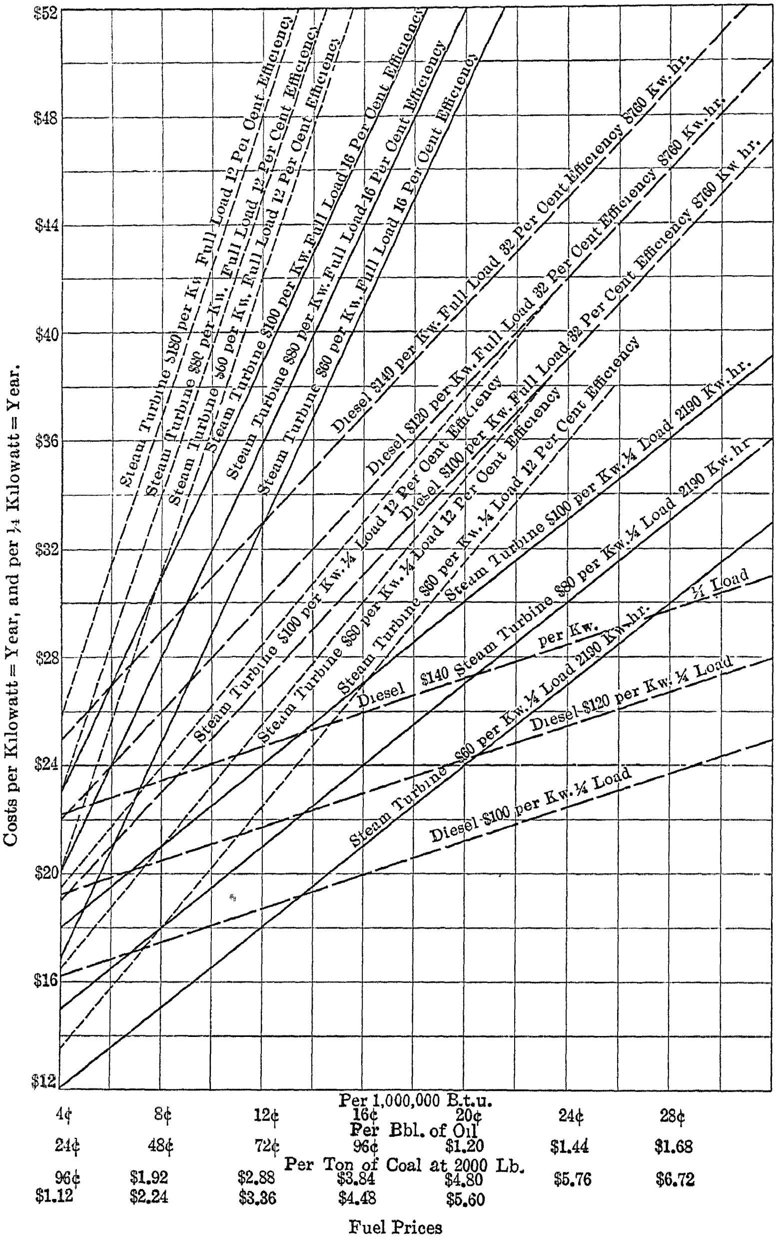
In comparing prices of different kinds of fuel, such as gas, coal, or liquid fuels, it is well to reduce all fuel prices to a common basis of absolute cost for 1,000,000 B.t.u. We have, then, for the cost of 1,000,000 B.t.u. transformed into mechanical work:

$$\frac{\text{Cost of 1,000,000 B.t.u.}}{\text{Over-all thermal efficiency of plant}}$$

and the fuel cost per brake horsepower-hour =

$$\frac{\text{Cost of 1,000,000 B.t.u.} \times 2{,}545}{\text{Over-all thermal efficiency} \times 1{,}000{,}000}$$

Knowing the thermal efficiencies of different prime movers, we can then arrive at the absolute fuel cost of each per brake horsepower-hour with different kinds of fuel. Thermal efficiencies of steam engines and turbines vary from 6 per cent. operating simple and non-condensing to 16 per cent. with large units, depending on size, and operating conditions; and of Diesel engines from 30 to 35 per cent., at full load. Thus, for a steam plant with a 12 per cent. efficiency, an absolute heat price of 12c. for 1,000,000 B.t.u. comes to 100c. for every 1,000,000 B.t.u. turned into effective mechanical work, whereas for a Diesel engine plant with, say 30 per cent. efficiency, 1,000,000 B.t.u. transformed into effective mechanical work would cost only 40c., and the respective costs per brake horsepower-hour would be 0.2545c. and 0.1018c.



Oil Costs Based on 1 Bbl. Containing 6,000,000 B. t. u.
Coal Costs Based on 1 lb. Containing 12,000 and 14,000 B. t. u.

FIG. 3.—DIAGRAM SHOWING COMPARATIVE POWER COSTS OF STEAM TURBINE AND DIESEL ENGINE PLANTS AT GIVEN PLANT COST, FUEL PRICE, AND LOAD FACTOR.

Where a supply of liquid fuel is available, the sound economic application of steam turbine and Diesel engine is determined by fuel prices, and load factor. The graph in Fig. 3 shows at a glance whether a steam turbine or a Diesel engine power plant is in place with a given fuel price and load factor. The abscissæ denote absolute fuel prices for each 1,000,000 B.t.u. These prices are translated into corresponding fuel prices per short ton (2,000 lb.) of bituminous coal having a fuel value of 12,000 and 14,000 B.t.u. per pound and per barrel of oil containing 6,000,000 B.t.u. The ordinates denote costs. The power costs represented by the curves include capital costs and fuel costs, but not operating labor, this being about the same for either type of plant, except for very large steam-turbine plants with units above 6,000 kw., which require less operating labor than Diesel engines and with which the latter can compete only under special conditions. The capital charges cover interest, amortization and maintenance, and are taken as 15 per cent. of the cost per kilowatt of installed plant capacity, which is for a complete, modern power plant with prime movers, switchboard equipment, and building, including necessary accessories, but without transformers or transmission line. The installation costs varying with size of plant and locality, three different costs per kilowatt of installed capacity are considered for each type of prime mover. For the turbine plant, \$60, \$80, \$100, and for the Diesel plant \$100, \$120, and \$140; 15 per cent. of these respective amounts are the capital charges added to the fuel costs, the curves marked "full load" showing the combined cost per kilowatt-year (8,760 kw.-hr.) of fuel and capital charges, the cost per kilowatt-year being shown in dollars in the ordinates.

The curves marked "quarter load" cover fuel cost for one-fourth kw.-year (or 2,190 kw.-hr) and the same capital charges as for full load; to arrive at the cost per kilowatt-year (8,760 kw.-hr.) *at quarter load,* the amount shown in dollars must be multiplied by four. This product divided by 8,760 will give the true cost per kilowatt-hour generated at quarter load.

Any other fractional load cost can be obtained by interpolation. An example will best illustrate the use of the graph: With a turbine plant costing \$60 per kw. of capacity and a Diesel plant \$100, the curve marked "steam turbine \$60 kw. full load 16 per cent. efficiency," will be seen to intersect the curve "Diesel \$100 per kw. full load 32 per cent. efficiency" at a fuel cost of 6c. per 1,000,000 B.t.u. and the cost per kilowatt-year (8,760 kw.-hr.) is \$21. Under these conditions, the costs being equal for both prime movers, it would not pay to use a Diesel plant, a turbine plant having certain mechanical advantages. To the right of the intersection of these two curves, *i.e.*, with an increase in the price of fuel, the Diesel engine is the more economical prime mover. Assuming a 20c. fuel cost per 1,000,000 B.t.u., we find a cost of \$35 per

kw.-year at full load for the Diesel plant, and $49 for the turbine plant (16 per cent. efficiency), the difference of $14 representing the fuel saving per kilowatt-year of the Diesel plant under full-load conditions. A plant of 2,000 kw. would save $28,000 per year. The corresponding curves marked one-fourth load will be seen to intersect at a fuel cost of 13½c. per 1,000,000 B.t.u. and a cost of $19.20 for 2,190 kw.-hr. (¼ load). The kilowatt-year at quarter-load conditions would cost $4 \times \$19.20 = \$76.80$. In this case, Diesel engines would only pay if the fuel costs were higher than 13½c. per 1,000,000 B.t.u. The divergence to the right between these curves indicates the fuel saving above capital charges for the Diesel engine (or, to the left, saving in capital charges above fuel saving, with turbines) per kilowatt at quarter load. At 20c. for 1,000,000 B.t.u., this divergence shows ($24–$21.20) a net saving of

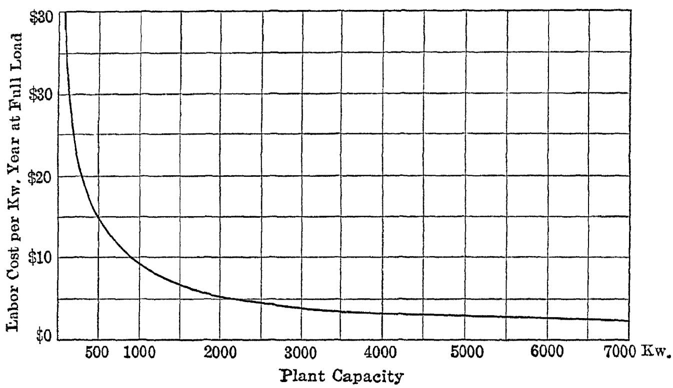


FIG. 4.—OPERATING LABOR COST PER KILOWATT-YEAR (8,760 KW-HR) AT FULL LOAD IN DIESEL ENGINE PLANTS.

$2.80 for the Diesel plant per kilowatt-year, for each 2,190 kw.-hr. per year (¼ load). For 75 and 50 per cent. load factor, the critical fuel prices can be ascertained by taking the difference between those at quarter and full loads, and adding two-thirds of this difference to the amount corresponding to quarter load to arrive at 75 per cent. load, and adding one-third of the difference to the amount for quarter load to arrive at 50 per cent. load. Any other critical fuel prices for other load conditions can be easily ascertained by interpolation.

It will be seen that curves based on 12 and 16 per cent. efficiency of steam-turbine plants are given. The higher figure is secured only with large units in sizes from 10,000 kw. upward. (These refer to the over-all efficiency of the entire power plant.) For smaller units, up to 2,000 kw., 12 per cent. represents good average practice. How a decrease in thermal efficiency favors the Diesel engine is strikingly shown by the

curves for the steam turbine, with 12 per cent. efficiency at full and quarter load.

To secure complete power costs, the cost of operating labor and lubrication must be added to the above costs. The labor cost per kilowatt-year varies greatly with the size of the plant. Fig. 4 shows average labor operating charges per kilowatt-year at full load for different-sized Diesel engine plants. This labor charge remains practically constant, regardless of load variations, so that with fractional loads the labor increases in inverse proportion. In turbine and Diesel plants here considered, the labor cost is about the same for either type of plant, as any increased attendance required by Diesel engines is offset by firemen, water tenders, and usual operating force of turbine plants. The cost for lubrication is considerably higher for Diesel plants than for turbine plants,

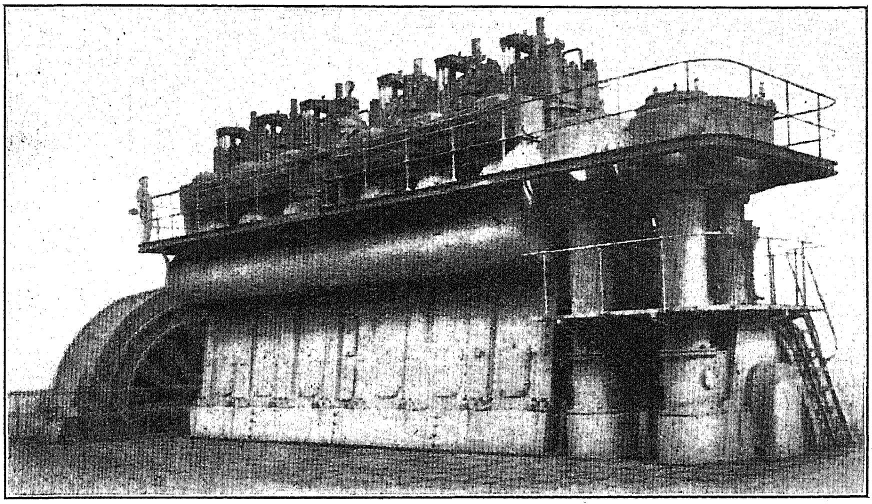


Fig. 5.—High-Duty, 4,000-Hp. Diesel Engine.

and amounts to from $2 to $4 per kilowatt-year, depending on the type of lubricating system used and the care of oilers. As a rule, a number of units are used in large Diesel installations, so that the lubrication expense is nearly proportional to the load, as with a decrease in load, units can be shut down. In turbine plants this greater expense for Diesel engines is partly balanced by the large amount of water required for condensing, which is 12 to 20 times greater than the supply needed for cooling Diesel engines. Only in plants with units in excess of 6,000 kw. have turbines decided advantages over Diesel engine plants.

Fig. 5 is a good example of a high-duty 4,000-hp. Diesel engine. In conclusion, Fig. 6 shows the present status of different types of prime movers, expressed in thermal balances.

It is hoped that the above notes will be found helpful to those having to select the most economical prime mover to meet certain conditions, either turbines or Diesel engines having a wide field of usefulness; a com-

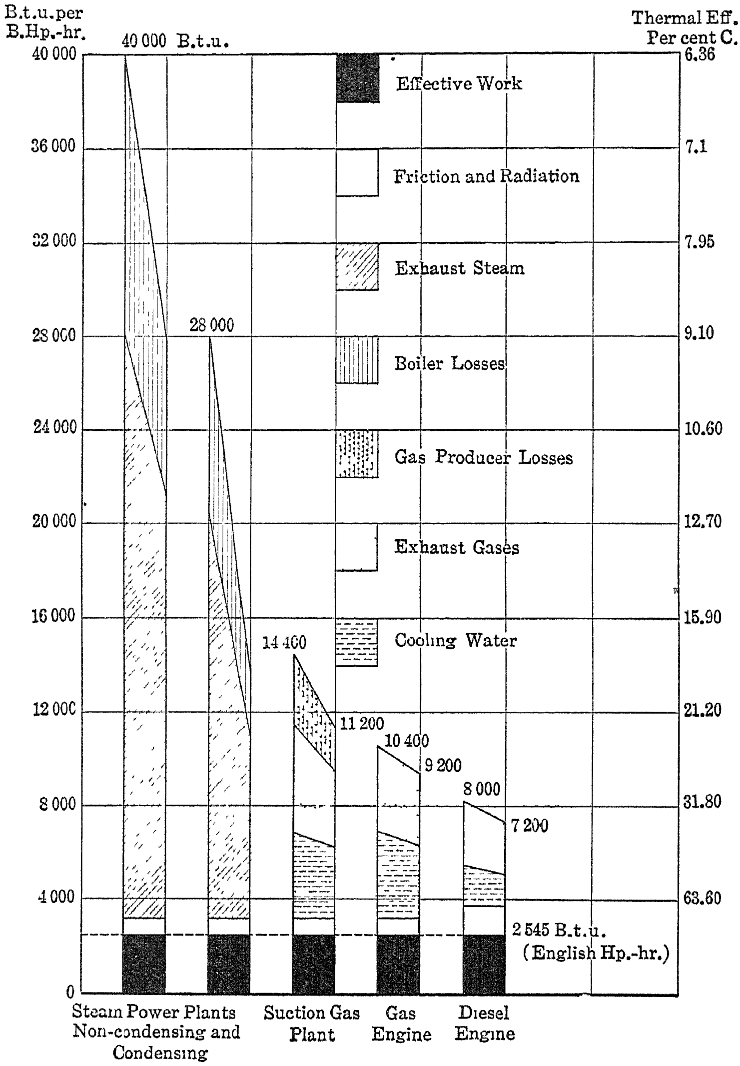


FIG. 6.—THERMAL BALANCE OF DIFFERENT HEAT ENGINES.

bination of both types often best meets particular conditions. The selection should be governed by the above discussed economic considerations, the most important of which will always be the fuel price and load factor.

Discussion

George W. Hawkins, Tucson, Ariz. (communication to the Secretary*).—The paper by Mr. Haas will no doubt be followed with considerable interest, as it covers the power-plant problem in quite a comprehensive way, including not only the usual operating costs but also the fixed charges on the investment, and for load factors ranging all the way from 25 to 100 per cent. and fuel oil ranging from $1.25 to $2 per barrel. The very fact, however, that his paper does cover such a wide range of conditions, makes it the more imperative that any conclusion drawn from his analysis be very carefully considered for any specific case, for in making such analysis the type of prime mover and size must be selected for some one condition which might not be the proper selection had some other condition been assumed as the basic one, resulting in an unfavorable comparison under all conditions except that condition assumed.

From the reasons given by the author for his selection of the large turbine units, it is evident he had in mind very high load factors, ranging from 75 to 100 per cent., and high fuel cost. Under these particular conditions, then, his comparison would be a fair one, but any comparison for different conditions than these, such as low load factors, or low fuel cost, would be very unfair, for in the latter case the plant should be designed along entirely different lines, resulting in different first cost and operating expense. That this criticism is a just one is evident from his tabulation of power costs, for according to that tabulation the lower the load factor the greater is the difference in operating costs in favor of the Diesel engines. For example: With oil at $1.25 per barrel, at 6,000-kw. load the difference in total operating costs between the steam plant and the Diesel plant is 0.217c. per kilowatt-hour in favor of the Diesel plant, whereas at one-quarter load this difference has increased to 0.342c. per kilowatt-hour in favor of the Diesel plant. This being contrary to the generally accepted opinion and also to his own statement that high load factor and high price of fuel are conditions which favor the Diesel engine, leads one to make a careful analysis of the basis of his figures.

Diesel Engines

Selection of Units

The full-load plant capacity is stated as 6,000 kw. in three units with one unit to serve as a standby. Inasmuch as it would be necessary to operate all of the units to get 8,000 kw., leaving no spare (which is not considered safe practice with Diesel engines), the comparison of the steam plant and Diesel engine plant at 8,000 kw. must be eliminated. With the

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steam plant as selected it would be possible to operate continuously at 8,000 kw., but not with the Diesel plant. The comparison, therefore, should be limited to an output of 6,000 kw., 4,000 kw. and 2,000 kw.

The author has selected units rated at 2,000 kw. each, these being of the vertical, single-acting, two-cycle type. The writer has carefully followed the development of Diesel engines in this country and does not know of a single installation having this size unit. Two of the largest Diesel engine builders take the position that, in the present state of the art, they would not attempt to put out units over 1,000 kw. Up to date the largest units in the United States are those at Tyrone, installed by Nordberg Mfg. Co., which are rated at 1,250 b.hp., or 800 kw. It is possible, of course, to build larger engines than the above by simply multiplying cylinders, maintaining the same horsepower per cylinder. This, however, does not reduce cost, except for generators, nor does it reduce operating expense. From the author's fourth reason (given on page 165) for the selection of this size unit, it is evident he figures on a unit having greater horsepower per cylinder. While it is true that in Europe units in excess of 2,000 hp. have been built in the two-cycle type, they are not considered, even by the manufacturers, to be out of the experimental stage; nevertheless, they have gained some experience in the manufacture and operation of these large units, which experience would not be of much value to American builders, as the possession of suitable patterns and instructions is not sufficient to insure successful construction of Diesel engines, practical shop knowledge of construction being just as important in turning out successful engines as proper design. This size unit is therefore entirely out of the question for adoption in mining plants or any plants where reliability is a prime necessity. Therefore, engines must be selected not larger than 1,000-kw. capacity. There would then be six 1,000-kw. units for the load, with two spares, giving a total installed capacity of 8,000 kw.

Cost of Plant

Based on the use of large units, the author has assumed a total plant cost of $90 per kilowatt. As smaller units are now being considered, the plant cost will have to be considerably revised.

The writer has seen estimates of costs of various plants ranging from $110 to $140 per kilowatt, depending upon size of unit, location, etc. The author, in a previous paper appearing in the *Engineering and Mining Journal* of Apr. 26, 1913, gives the cost for a plant of 1,400 kw. as $135 per kilowatt, this being based on 450-kw. units. Owing to the fact that the units under consideration here are larger and the plant itself of larger capacity, the writer believes that $110 to $120 per kilowatt would be considered a fair estimate of the cost of this plant, this cost including not only the engines and generators, but the piping, foundations, oil

tanks, crane, building, jacket water, cooling system, etc. The installed cost of this plant would then be between $880,000 and $960,000 as against the author's first cost of $720,000.

Fixed Charges

Fixed charge of 12 per cent., consisting of interest at 6 per cent., amortization at 6 per cent., has been assumed by Mr. Haas. The amortization, or sinking-fund figure, is based upon 15 years' life of plant, sinking fund bearing 4½ per cent. interest. The life of the Diesel engine is an unknown factor, the present engines having been developed within comparatively recent years. The proper amount to figure for sinking fund is therefore merely an intelligent guess. Some Diesel engine builders figure that there should be an allowance made to cover the entire replacement of the Diesel engines every 9 or 10 years.

As against this, the life of the steam plant is well known. Steam plants are in good operating condition 20 to 25 years after installation, and the steam turbine has been perfected to such a degree that it is not likely that there will be any radical departure from present types for a good many years to come. Whatever figure is used for the assumed life of the Diesel engine plant, the life of the steam plant should be figured at least 50 per cent. longer. If 15 years is assumed for the Diesel engine, resulting in a sinking fund of 6 per cent. with money reinvested at 4½ per cent., 25 years should be figured for the steam plant, giving a sinking fund of 2.25 per cent., or 3.75 per cent. greater fixed charges for the Diesel plant than for the steam. Considering that the first cost is about 33⅓ per cent. higher than assumed by the author, the increase in this item of fixed charges is a very material one.

Fuel

Fuel cost is, of course, the largest item of expense at high load factors, and any wrong assumption in this item would affect results very materially. The author has used a figure of 0.64 lb. of oil per kilowatt hour in arriving at his full-load fuel cost, and in fact, his fractional-load fuel cost. This figure is about what builders usually guarantee for four-cycle engines, and it may be obtained on acceptance tests, but there is of necessity always a falling down in actual operation from the test results, due to the method of operation, the way engines are kept up, etc. The fuel consumption, therefore, under normal operation, even when the plant is new, would be at least 5 per cent. higher. Owing to the excessively high pressure used in Diesel engines, any slight wear of cylinder would reduce economy very materially due to the enormous leakage possible, and hence, the average economy over the life of plant

is likely to be considerably poorer than for the first few months of operation.

Owing to the very short time the Diesel type of plant has been in operation, it is impossible to tell just what this falling off in economy would be from operation the first few months, but that there will be such a falling off is certain. However, this is not being considered in this analysis. Engines of this large size, however, would probably be of the two-cycle type, which is not as economical as the four-cycle. The author makes the statement in the former paper referred to that the two-cycle engines have fully 10 per cent. higher fuel consumption than the four-cycle engines, making the actual operating economy 0.736 lb. per kilowatt-hour. His fuel consumption,therefore, should be 15 per cent. higher than he has figured.

Further, he has assumed that plant would operate on ordinary fuel oil running 320 lb. per barrel. Diesel engine manufacturers are willing to guarantee operation with heavy-gravity fuel oils, but except in a few instances, even where such guarantees have been made, the owners have after a short trial gone to the lighter-gravity oils, owing to the heavy cost and trouble of keeping engine in running condition. To be perfectly safe, therefore, in deciding type of prime mover, one should not base operating costs on the use of heavy-gravity oils, and as the lighter oils run from 15 to 20c. per barrel more than the heavy oils in common use in steam stations, and run considerably lighter per barrel, as low as 285 or 300 lb., there is the possibility of further increase in the cost of fuel oil over that assumed by the author. The additional fuel cost at full load would therefore be at least $20,000 greater than given by the author, and may run as much as $46,000 greater if the lighter-gravity oils should be adopted for the Diesel plant. In this analysis, however, it is assumed that fuel oil can be used.

The above applies to full-load operation. At fractional loads there will be a falling off in economy of the Diesel engine in the same way that there is for steam turbines. Theoretically, it would be possible to get full-load economy at any of the fractional plant loads due to the number of small units, enabling the units in service to be operated at practically rating. This is not practical working condition, however, as Diesel engines have a fixed overload, and on a swinging load it would not be possible to have in service just the proper number of units theoretically required. This would mean that the units in service would be operated at fractional loads, in which case the economy would drop off quite materially, the same as with any type of prime mover.

Maintenance and Lubrication

The next item of importance is maintenance. The author has assumed maintenance as 1 per cent. of the plant cost. This cannot be

based on actual operating costs in this country for any lengthened period, as no plants of any size have been in operation long enough to give any reliable records. It would be like assuming the average maintenance cost of an automobile from the records of the first few months of its operation. On the other hand, it may be this is based on operation in European countries. However, considering the difference in price of labor, price of material, class of labor, etc., it is practically impossible to make proper adjustment to arrive at a figure for operation in the United States. The safer plan would be to investigate maintenance costs of actual plants in the United States, increasing these items to cover average maintenance over the period of the life of plant, and then make any adjustment necessary for more favorable conditions in this plant.

In the author's article in the *Engineering and Mining Journal* he assumes a figure which is 3 per cent. of the first cost of the plant for maintenance, making the statement that the figure is based on prolonged cost records of Diesel engine power plants. The writer has in mind two plants of fairly good size where the first 2 years' operation was even higher than this, and it is safe to assume that the maintenance cost would increase as the plant gets older.

The cost of lubrication is one that varies widely, depending upon the design of engine, care and operation, etc., varying all the way from one-fourth to one-eighth of the fuel bill, so that the figure given by the author can be considered as a general average, although it might mount up to a considerably higher figure.

Labor

The item of labor, considering the greater number of units, will be higher than assumed by Mr. Haas. In addition to the engineer and his assistants and electrical operators, there should be one oiler for two engines, which would make nine oilers per 24-hr. period. This adds a considerable amount to the labor rate.

Steam Plant

Selection of Units

Mr. Haas says that his reason for selecting 6,000-kw. turbines is the superior economy of the 6,000-kw. turbine over, say, a 3,000-kw. turbine, which justified the very much heavier first cost. This is the proper way to select size of unit, but it can apply only to one particular assumed condition. If the plant is to run at 100 per cent. load factor the superior economy of the 6,000-kw. unit will offset the fixed charges on the greater investment, but if the plant is to operate at one-half, one-third or one-quarter load, the selection of a 6,000-kw. unit may be a very bad one, for the total operating cost might figure out much lower

with a smaller unit because of superior economy at the load considered and lower first cost due to smaller installed capacity.

Cost of Plant

Assuming that his selection of 6,000-kw. turbines is the proper one, his plant cost is about right, but if plant were to operate at, say, one-half load or one-third load, an investigation of the total operating cost would probably lead to the selection of two 3,000-kw. units to carry the load with one spare, or a total installed capacity of 9,000 kw. against an installed capacity of 12,000 kw., resulting in the plant costing probably 25 per cent. less than assumed by Mr. Haas.

Fixed Charges

As stated above, if life of the Diesel plant is considered as 15 years, the life of the steam plant ought to be, for fair comparison, assumed at 25 years, resulting in 3.75 per cent. lower fixed charge for the steam plant than for the Diesel plant.

Fuel

While the assumed steam-plant economy checks with average practice, it is not by a good large percentage as high as is actually being obtained in some recent plants under daily operating conditions with units of approximately the same size as assumed here, this increase in economy being obtained by advance in boiler design—allowing increased steam pressure and higher superheat—the use of steel-encased boilers, the use of automatic oil firing—giving better furnace regulation—the use of properly designed cooling system, and the advance in condenser design allowing higher vacuums, etc. These are not merely possible improvements in steam-plant design, but they have actually been incorporated in some existing plants, resulting in the high economy stated. Even these economies can be exceeded where plant is installed at seaboard where colder condenser water is available, giving higher vacuum, and consequently better economy of prime mover.

Further, it is just as easy to burn a lower-gravity oil as to burn lighter oils, the low-gravity oils running greater weight per barrel and costing the same, or slightly less. On the other hand, it would not be wise to use this low-gravity oil for Diesel engines.

The fuel item, then, could easily be as much as 20 per cent. lower than assumed by the author, which would mean practically $52,000 smaller yearly fuel bill at rated load. When operating at the lower loads, his fuel bill is still further off because, as explained above, for a plant operating under these conditions one would not think of installing a one-unit plant, and hence, although the smaller unit would have a lower economy,

it would be operating at full load and therefore at better economy than the large unit at fractional load. For this reason, the fuel items at fractional loads are far too high, resulting in an unfair comparison.

Maintenance, Etc.

The maintenance item of a steam turbine plant is one of the smaller items of cost, as it simply means the expense of cleaning boilers, small miscellaneous apparatus, and an occasional repair, so that the figure Mr. Haas has used is exceedingly ample.

The water bill, of course, is much higher than for Diesel plants, but it is not such an item as to have very much weight in the decision of the type of plant to be installed; that is, as far as the cost of it is concerned. The question of sufficient supply is an entirely different matter.

The labor cost of operating a steam plant is less than for a Diesel, as a steam plant requires very little attention compared with a Diesel engine; and with an oil-fired plant and automatic system of oil firing, the boiler room labor is cut down to a minimum.

Diesel and Turbine Plant Diagrams

In addition to the tabulated analysis, Mr. Haas submits diagrams, Fig. 3, showing diagrammatically the relation between the operating costs of steam turbine plants and Diesel engine plants at various efficiencies, fuel costs and load factors. These diagrams are quite misleading in that they do not include all the operating costs. The reason given by the author for omitting these is that the remaining operating costs are either of small moment or practically the same for both types of plants. This is not true, as both maintenance and lubrication of the Diesel plant are excessively high and have no parallel in the steam plant. He states that lubrication alone varies from $2 to $4 per kilowatt-hour. The maintenance item will also run about the same rate, so that if these items were included it would make quite a difference in the comparison.

Further, the quarter-load comparison is unfavorable to the steam plant on account of the efficiency assumed. Checking back from some-of the figures, it is found that the economy at quarter load is reduced about 50 per cent. over what it is at full load. This would probably be true if a one-unit plant were used, but as explained above, under such operating conditions a two- or three-unit plant would be selected, and the economy would then not be more than 15 per cent. to 20 per cent. poorer than the plant operating at full load.

Conclusion

The writer has made some figures corrected along the above lines which show that the operating costs of both types of plant are materially

different from those given by the author. At 6,000-kw. output, which is considered full rating of the plant, the total operating cost per year for the Diesel engine plant is $345,000 and for the steam plant $305,000, making 0.657c. per kilowatt-hour for the Diesel plant and 0.58c. per kilowatt-hour for the steam plant. One hundred per cent. load factor, however, is only a theoretical condition and is seldom, if ever, obtained in practice, if units are properly selected when installing plant. Three-quarters load, or 4,500-kw. output, represents more nearly an operating condition, but under this condition it is still more favorable to the steam plant, the cost per kilowatt-hour for the Diesel plant being 0.76c. and for the steam plant 0.65c.

The above is based upon using the same grade of fuel oil for both the steam plant and the Diesel. If lighter-gravity oils were used for the Diesel, the operating cost of the Diesel plant would be still greater, and this is always a possibility as this point is not yet beyond question.

Summing up the above arguments, the author's paper is shown to give an untrue comparison of the relative operating costs of the two types of prime mover, on account of:

1. Selecting Diesel engines of far larger rating than have ever been put into successful use.

2. First cost of plant too low, resulting from this selection of units.

3. Assuming actual operating economy equivalent to builders' guarantees and not reducing this to cover results obtained in actual operation.

4. Maintenance too low, being merely an assumed figure and not checking with published records.

5. Economy of the steam plant has been assumed far lower than is actually being obtained in the latest turbine plants.

6. Fixed charges on steam plant being too high in comparison with fixed charges assumed for Diesel plant on account of same life being assumed for both steam and Diesel plants.

7. Fractional-load economy on turbine plant is erroneous on account of selecting units on the basis of 100 per cent. load factor and then operating them at fractional loads, instead of, in the case of fractional load, assuming different size units which would result in better economy and lower first cost.

While the figures used by the writer for Diesel engine operating costs are very much higher than the author's, they cannot be considered extreme for they have been compared with published records of actual operating costs, and in each case lower figures have been assumed on account of the units being larger and representing the latest development in Diesel engine design. It must further be borne in mind that in a comparison of this kind one is comparing the well-known reliability and well-known cost of operation of a steam turbine plant with a type of

plant which is entirely untried for plants of this large capacity; consequently, the operating costs are entirely a matter of conjecture and in using figures somewhat better than published records of such plants, one is going as far as it is safe to go.

Aside from the question of operating costs, there are other considerations which are of vital importance in the selection of prime mover for a mine plant, where any shutdown of mine due to power-plant troubles would be very serious, hence the question of absolute reliability of operation, the ease of getting sufficient supply of skilled labor familiar with the operation of the type of prime mover selected, the ease or difficulty of making repairs or getting repair parts, the suitability of the plant for the use of other fuels beside oil, are of prime importance. All of these considerations will tend toward the adoption of the steam turbine plant rather than the Diesel plant, even if the operating costs as estimated did show up favorably to the Diesel engine. All such questions are already answered in the case of a steam turbine plant, where they are entirely problematical in Diesel engines.

Considering all these vital points, the Diesel engine must show a tremendous margin on paper in operating costs over the steam plant before the risk is justified in mine power plants.

In conclusion, the writer would refer to the statement made by H. J. Freyn, the well-known authority on Internal Combustion Engines, appearing in the *Transactions of the American Society of Mechanical Engineers* in 1911, as follows:[1]

"Since the Diesel engine owes its existence primarily, in fact almost exclusively, to its unsurpassed fuel economy, it is not surprising that its advent and development have had a singular economic significance in Germany and France and on the European continent in general, where fuel is very expensive and the standard of manufacture high, whereas in England and more particularly in America, where excellent cheap fuels abound, while skilled labor is scarce and dear, there is manifestly not the same inducement for the introduction of very costly although thermally highly economic machinery."

Also—

"Broadly speaking the outlook for the oil engine of medium size in this country seems good, perhaps not so much in the immediate future as later on after more knowledge of the advantages of the oil engine is disseminated and its good features are demonstrated; but over-enthusiasm should be carefully guarded against, since, as mentioned before, economic conditions in this country are materially different from those abroad, where the oil engine seems to have entered upon a triumphal career."

Additional discussion of this paper on p 952.

[1] Vol. 33, pp. 921 to 923.

Motor Truck Operation at Mammoth Collins Mine, Shultz, Ariz.

BY WILBERT G. McBRIDE,* B. S., LONG BEACH, CAL.

(Arizona Meeting, September, 1916)

Two Alco 3½-ton motor trucks were used by Young Bros. while operating at the Mammoth Collins mine at Shultz, Ariz. One was equipped with an oil tank holding 1,075 gal. and was used for the transportation of "tops." The other was fitted with a stake body and used to carry machinery, wood, rails, pipe and all classes of miscellaneous supplies. The bodies were made of oak with maple flooring and were attached to the frame of the chassis by U-bolts, to avoid drilling the main members of the frame.

Most of the hauling was done from Tucson, a distance of 47½ miles. During the first 3 months, part of the road was in very bad condition and the tire cost was excessive. After this part was repaired the road was in fair condition, but never good. There were no excessive grades or bad sand, but wagon ruts, too narrow for the truck wheels and of a different gage, caused heavy tire loss; while chuck holes, sharp curves and stones, both imbedded and loose, were objectionable features. During wet weather the trucks could not get sufficient traction to climb some of the hills and were likely to stick in the mud in certain places, so that no attempt was made to run them unless they were on the road when the rain started. This lost time amounted to about 5 per cent. of the total, but, whenever possible, it was utilized in making minor repairs.

The price of gasoline was from 17 to 21 c. per gal. Rubber tires were used throughout. Drivers were paid $4.50 to $5 per shift, and a return trip to Tucson was counted as two shifts even when made in one day. Drivers were provided with a room in Tucson and were paid for all time lost due to causes beyond their control. Trucks were loaded one way only.

Speedometers were placed on both trucks but the excessive vibration soon caused them to fail. For this reason, and because no account was taken of the distance covered in picking up a miscellaneous load or in other minor ways, the mileage given is under the actual distance traveled. Some of the weights had to be estimated, but care was taken to have the number of ton-miles low rather than high, to avoid underestimating the costs. The cost of hauling from Tucson to the mines was $12 per ton

* Consulting Engineer.

with the trucks, while the best possible team price was $15. Teams made one return trip a week, while the truck regularly made one in two days and could always, and many times did, do it in one day. The loss of time due to wet weather would be about half as much with teams as with trucks.

The table of detailed costs given below covered the period from Aug. 21, 1913, to Aug. 15, 1914, the only time in which the trucks were continuously employed. From Aug. 15, 1914, to March 30, 1916, the trucks were used intermittently, but the figures for this period have been excluded as not being representative. If included, they would somewhat lower the cost per ton-mile. Just prior to the close of the period covered by the cost figures, the trucks were overhauled and put in good condition; new rear wheels were put on and new tires secured. The cost of all this was charged to operation. Allowance for extra tires on hand would reduce the cost per ton-mile approximately ¾ c., leaving a net cost of about 25c. With loads on the return trip this cost per ton-mile would be lowered at least 40 per cent.

Motor Truck Operating Data

Total distance traveled by trucks	23,000 miles
Total work done by trucks	42,700 ton-miles
Average distance covered per gallon of gasoline	4 5 miles
Average distance covered per gallon of lubricating oil	128 miles
Average speed, loaded	7 miles per hour
Average speed, light	7.8 miles per hour

Details of Costs

	Total Cost	Per Cent. of Total	Per Truck-mile	Per Ton-mile
Wages of drivers	$2,623.32	23 91	$0 1141	$0.0614
Wages of helpers	286.50	2 62	0 0125	0 0067
Repairs, labor	581 74	5 30	0 0253	0.0136
Repairs, lost time	156.15	1 42	0 0068	0.0037
Oils, grease and waste	379.17	3 46	0 0165	0 0089
Gasoline	1,610 49	14 68	0.0700	0 0377
Tires	2,445.75	22 30	0.1063	0 0573
New parts	515.08	4 69	0.0224	0.0121
Miscellaneous supplies	348.82	3.18	0.0152	0 0082
Incidental expense	226.21	2.06	0 0098	0.0053
Depreciation	1,796.80	16.38	0 0781	0.0421
Total	10,970.03	100 00	0 4770	0 2570

The advantages of the motor truck over the team and wagon are many—increased speed, ability to work 24 hr. per day when necessary, and lower cost on long hauls—but its adoption by the mining industry

has been slow. Where trucks are used around mines they are usually driven by cheap, inexperienced men, the upkeep and repairs being turned over to the regular mine mechanics. It would be equally good practice to employ a timber framer to make a dining-room table. Just as the niceties of cabinet making are unknown to the timber framer, the exact adjustments and fine workmanship of the high-speed engine and transmission gears of a motor truck are beyond the ken of the mine mechanic, one of the least skilled of his class. If there are enough motor vehicles at the mine, the master mechanic probably turns the work over to one or two men who, in time, become indifferent auto-mechanics, but in the meantime the cost of maintenance soars and often the trucks are condemned. The aim of the makers of all motor vehicles is to secure the maximum of strength and power with a minimum of weight and size. To do this, high-speed engines, the best of materials and the finest of workmanship are employed and parts are reduced to the least possible weight consistent with strength and durability. This is just the reverse of the ordinary American mechanical practice in which reliability is secured by slow speed and large size, the amount of material used and the space occupied being minor considerations. It is, therefore, unreasonable to expect the mechanic trained in one school to understand immediately and adapt himself to the ways of the other. It must also be remembered that no other machine is given the hard use and necessary abuse that a motor truck receives. The road vibration, alone, will loosen nuts and rivets which, if not attended to in time, will cause serious trouble. Where only one or two trucks are used, the drivers should be competent mechanics and should be held responsible for the maintenance of their machines. Where several are used, they should be under the direct supervision of a thorough truck mechanic who is held responsible for operation and given entire control of the drivers and repair work. His constant care will detect and remedy many incipient defects and prevent expensive and annoying breakdowns. With the exception of the time required for periodic overhauling, he should be able to keep the trucks in almost continuous service. This will make possible the employment of cheaper drivers without undue damage being done to the machines.

Motor trucks should not be installed without careful consideration of the roads to be traveled. The difference between the cost of motor truck and team hauling is largely controlled by the quality of the road, and on really bad roads the motor truck is decidedly the more expensive. Many roads are fatal to truck haulage, and considerable experience is required to decide this question without an actual test of some duration. Excessive grades are to be avoided, especially long ones. The ordinary truck will pull over a short 20 per cent. grade with ease, but will give great trouble on a long one of half that rise unless special cooling arrangements are made. Grades greatly increase the tire and gasoline consump-

tion and decrease the life of the machine. Rocky roads, particularly when the rocks are sharp or loose, are very hard on tires. Deep sand is difficult to cross, and for this class of road the caterpillar tractor and the four-wheel-drive truck have distinct advantages. Trucks which drive on the rear wheels only cannot operate in heavy sand. Narrow or rutted roads are objectionable for the larger-sized trucks because they throw all the weight on one of the rear dual tires from time to time, and this overloading is injurious to the rubber. Fairly deep streams can be crossed, but mud is an absolute barrier except to the caterpillar type of tractor. Few dirt roads will stand up under a 7-ton truck, but those of 4 tons, or under, do less damage than the ordinary freight wagon.

Unfortunately, trucks are not designed to suit mining conditions. At Shultz we found it necessary to cut down the gear ratio, increase the size of the wheels and tires and add bumper or auxiliary springs. Had the grades been steeper it would have been necessary to increase the cooling capacity.

For long hauls the motor tractor will probably replace the motor truck. It will operate at a lower cost because the load will be carried on iron tires, and, as the table of detailed costs shows, the rubber tires account for 22.3 per cent. of the total. Tractors travel more slowly than motor trucks, but the tonnage hauled in a trip is much greater. They are also easier on roads, as the load is distributed over several trailers. By using extra trailers, loading and unloading can be done while the tractor is on the road.

The make of a truck is not as important as the care it receives. Almost any standard make will do good work if given careful attention, but none will be satisfactory if not well cared for. Economy should not be sought in the lubricants used; the best oil is none too good for a motor truck. Overloading should be scrupulously avoided. A truck may be made to carry many times its rated load without breaking down but the damage is none the less real because not immediately apparent. High speed, particularly if the road is rough, should be avoided, since it subjects the machine to excessive strains and vibration. Most trucks are now equipped with speed governors, but these are easily tampered with and must be carefully watched. When they are not used, the drivers should be carefully instructed as to the speed limits and compelled to respect them.

Distillate and "tops" are now successfully used on trucks, by the application of a special carburetor. Their use should effect a material saving in the gasoline cost which now amounts to almost 15 per cent. of the total. "Tops" usually sell for 30 to 35 per cent. and distillate for 50 to 60 per cent. of the price of gasoline. With a properly designed carburetor, the available power in the lower-grade fuel will be about the same as in the gasoline, but the carbon deposition will probably be somewhat greater.

Mine-Fire Methods Employed by the United Verde Copper Co.

BY ROBERT E. TALLY,* B. S., JEROME, ARIZ.

(Arizona Meeting, September, 1916)

UNDERGROUND fires have been common in the mines of the United Verde Copper Co. for the past 22 years. The first fire started in the 300 Hampton stope in the fall of 1894, following a cave in that orebody. The soft and highly pyritic nature of the ores is responsible for most of these fires. Since ore characteristics, methods of mining and ventilation, have an important influence on mine fires, they will be described in this paper.

Orebodies

The ores are mostly chalcopyrite except on the upper levels where considerable chalcocite, black oxides and other secondary minerals occur. The orebodies are in the form of lenses and vary in size. Some are small, while others are several hundred feet in length by a hundred or more feet in width. These orebodies extend, in some cases, from the surface to the lowest levels of the mine and usually at a steep angle of inclination.

There are three different classes of ores, classified in accordance with the gangue rock. The copper in all cases, except as above specified, is in the form of chalcopyrite. The largest and most important of these reserves are in a pyrite gangue. Next in order of importance are the schist ores and lastly the quartz porphyries. The pyrite, locally termed iron ores, average from 30 to 42 per cent. sulphur. The schist and quartz porphyries vary from 3 to 25 per cent. sulphur.

Mining Methods

All the orebodies were worked by the overhand square-set method of mining prior to 1908; since then the cut-and-fill, shrinkage and mill-hole systems have been largely employed.

But 15 per cent. of the total tonnage now extracted is from timbered stopes. Most of this is from the upper levels where the ore is soft, heavy and more or less broken to the surface. Large reserves of these ores are being conserved for more efficient methods of mining.

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* Superintendent of Mines, United Verde Copper Co.

There was no choice in the past with reference to the methods of working the upper levels because the smelting plant and other surface buildings were directly over the orebodies. Only such places were worked as were considered necessary and by methods which caused as little settling as possible. In most of these places it was necessary to use spiling and to fill to the roof. Bulkheads were used on the levels to retain the gangways and were carried up through the stopes for ore chutes.

Top slicing could not be used on account of surface settling and because this method is unfavorable, in heavy sulphide orebodies, for the control of mine fires.

The smelting plant and other surface interferences have now been removed and it is planned to start steam-shovel work in the near future.

History of United Verde Fires

The first fire of 1894 was caused by spontaneous combustion, following a cave. It was not extinguished and the district was bulkheaded from the remainder of the mine. The soft and broken nature of the ground in this district was such that the smoke and gas worked its way over the orebody on the several different levels and cut off a large productive area of high-grade ore.

Later, another serious fire occurred in the Chrome or quartz-porphyry orebodies, which extended over a large area and cut off production from this district.

A third fire followed, cutting off the middle district which lies between the Chromes and Hamptons. Many smaller fires occurred in the meantime, which were extinguished. There has never been a serious fire in the schist orebodies which join the Hamptons on the south, yet there is always danger in that or any other heavily timbered district, where the ore carries any appreciable amount of sulphur.

Fire Stopes

Many attempts were made to extinguish these fires and recover the large reserves of high-grade ore. The districts were first flooded with water but, owing to the large area involved and the soft, broken-up condition of the ground, this method was a failure. Sufficient air always found its way through fractures to feed the fire.

The next attempt to extinguish the fire was by CO_2 gas. A gas-generating plant was installed, pipe lines were laid and large volumes of gas were forced into the district, but without effect. Steam was then tried and was likewise a failure. Water, steam or CO_2 gas will readily extinguish a fire if the district in which the fire occurs can be tightly sealed up and the extinguishing agent brought into contact with the fire in

sufficient quantities. Fractures to the surface and underground workings outside the fire district made an impossible condition to seal up.

In 1905, the Plenum system was tried on this district with success. Actual combustion was extinguished and the latent heat reduced to a temperature of about 120°. This work was planned and directed by

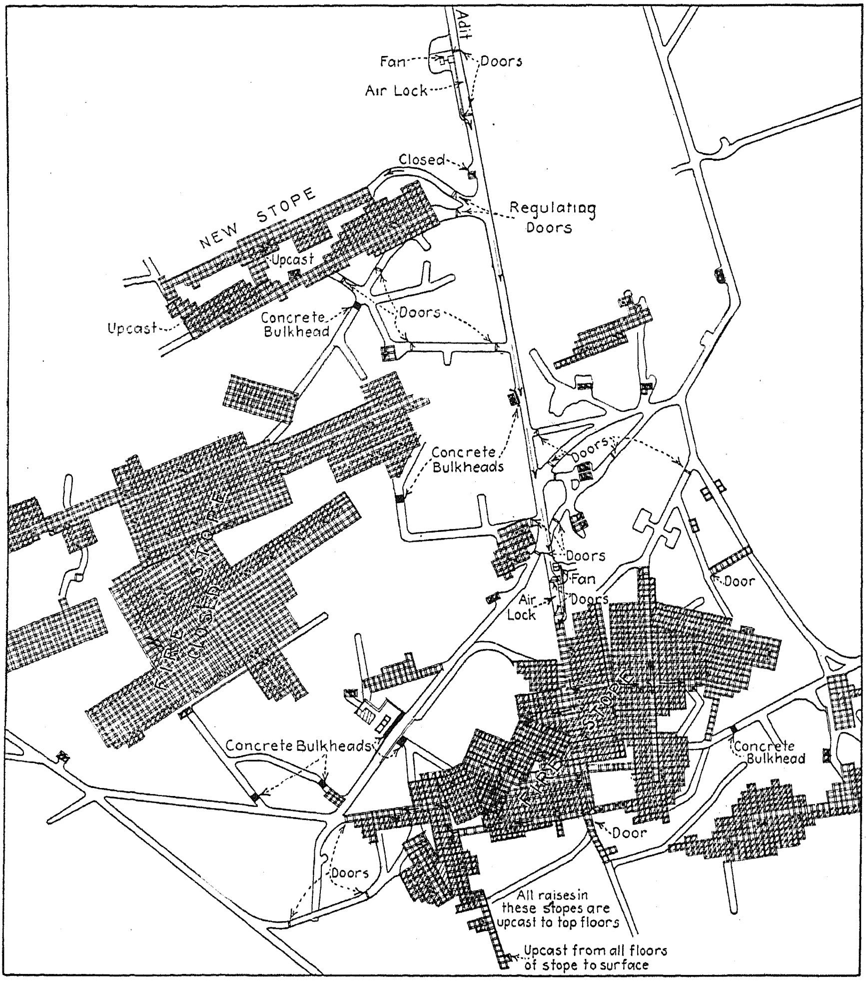

FIG. 1.—PLAN OF FIRE STOPES ON 300 LEVEL SHOWING METHOD OF VENTILATION.

Joseph J. Shaw, a mining engineer from the Mountain Copper Co. of Keswick, Cal., where this system was successfully used. It has been used here continually since its inception and consists in forcing the air under a pressure, varying from 2 to 5 lb., into the fire district. The air pressure varies in accordance with the gas pressure and must be sufficient

to keep back the gas and to cool the ground, so that work can be accomplished.

When a bulkhead, in a connection leading to a fire stope, is opened there is invariably an outward pressure and rush of gas or smoke, which must be overcome. The Plenum system consists in forcing the air against this gas at a pressure slightly greater than that of the gas.

Fig. 1 shows a large fire stope on the 300 level, with doors for controlling the pressure, and the method of ventilation. The air is furnished from the adit and forced under pressure into the fire district. All chutes and manways are upcast, and the spent air and gases outlet to the air raise through connections on the different floors.

The installation used with this system consists of a No. 8 double-inlet Sirocco fan, direct-connected to a 50-hp. variable-speed General Electric motor; also the necessary airways, doors, etc. The air from the fan must be under perfect control in order to regulate the gas. This is accomplished by doors, as shown in Fig. 2.

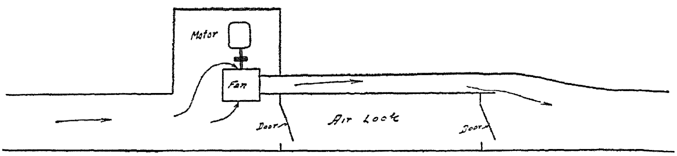

FIG. 2.—SKETCH OF FAN STATION

An airway about 3 ft. square and from 50 to 200 ft. long is connected to the outlet of the fan. Doors are placed in the tunnel at both ends of this airway. These doors prevent the fan from drawing its own air, for when one door is open the other must be closed. All workings between the fan and the fire district must be closed off by doors; otherwise the air pressure will be reduced and trouble result from gas. Two doors are always used so that one will remain closed when the other is open. For electric haulage these doors should be from 100 to 200 ft. apart. The doors are made of 2-in. grooved lumber, and concrete frames. The airways are made of lumber or concrete.

With a variable-speed motor and the proper system of doors, it is an easy matter to increase or diminish the air pressure in order to meet any new conditions within the fire district.

When the bulkheads to this district were first opened, the ground was red hot. The roof and walls, where opened, were aglow. All timber was burned and the soft sulphides, where exposed, were burning. The continual blowing of air on this hot mass gradually cooled it, so that within a few weeks the actual extraction of ore was under way.

The timber often took fire before it was blocked, making it necessary to keep it well sprinkled with water. The air reduced the temperature

rom 1,200° F. to 120° F. in about 6 weeks. The conditions gradually mproved until the average temperature in the fire stopes was 100° F. During recent years a system of ventilation has been introduced whereby the temperature has been reduced to about 75°, except in places that are above the line of ventilation.

Formerly the air was forced into these stopes and found an outlet through fractures into the other workings and to the surface, with the result that none of the upper levels were free from gas.

A system of air raises was driven in the foot wall to the surface and connections from these raises to the stopes were made at different elevations. Doors were placed in these connections and the air outlet thereby controlled. If an outlet is too free, it will not only release the pressure and cause gas, but, by suction, will also cause fire. These raises were driven in ground that did not require timbering. Cast-iron sets were used where necessary in the connections from the raises to the stopes. Timber should be avoided if possible in airways, since there is a constant danger of fire in these places.

These stopes have now been opened for 11 years and fire occurs on an average of about once a month in a rather extensive area of old timbered workings. It is therefore important to have an efficient fire-fighting organization and to have good control of the ventilation. This is an important factor in connection with mine fires. In the first place, every preventive measure must be used, and, in the second, the ventilation must be under absolute control in order to extinguish the fire.

An excess of air increases the flames in the same manner that an open draft increases the fire in a stove or furnace. Air must be furnished only in sufficient quantities to carry away the smoke so that the firemen can get at the fire. Where the smoke is thick it is impossible, even with helmets, to handle the fire satisfactorily. Many unsatisfactory results in connection with mine fires are due to these causes.

Causes of Mine Fires

Probably 90 per cent. of all underground fires in pyrite or heavy sulphide ores are caused by spontaneous combustion. In the fire district of this mine, many fires are due to red-hot dust dropping on the timbers, from fractures in ground above. Some fires are from friction due to caving, some from defective electric wires or cables; others from burning powder in contact with timber, and a certain number from incendiarism and carelessness with candles, lamps, etc.

Fires from spontaneous combustion usually occur in the interior of a gob or filled stope and are due to the oxidization of fine sulphides in contact with timber. Very small amounts of air are necessary for spontaneous combustion and this air works its way through the filling along the ine of bulkheads, timber or walls, where it comes into contact with sul-

phide dust. In the process of oxidization this produces sufficient heat to start a fire. This theory has been proven here by experiments. Heavy sulphides should not be used for filling in timbered stopes. Considerable fine sulphide ore is lost in the fire stopes or in any other heavy ground where the square-set method of mining is used and where the floors are kept filled to the roof.

Prevention of Mine Fires

Careful consideration should be given toward making heavy sulphide mines as fireproof as economy will permit.

Main hoisting shafts in such properties should be constructed of concrete or other fireproof material. Where timber is used, a sprinkling system should be installed.

Mine methods should be developed in which timber will be eliminated as far as possible.

Heavy sulphide material should not be used for filling in the timbered stopes. In these places fine waste should be used and sprinkled so that it will pack and make conditions unfavorable for fire.

In fire stopes or stopes adjacent to the fire district, the timber next to the walls or ends should be removed.

Careful attention should be directed toward ventilation so that the temperature will be unfavorable for fires to start.

In hot places the timber should be kept damp by sprinkling with water. Water should never be put on red-hot or burning ground, as this will invariable cause an explosion, one of the great dangers in handling mine fires.

Water and air lines, hose and hose connections, and all tools used in connection with mine fires should be kept convenient and in good working condition where fire is liable to occur.

Helmets, pulmotors or lungmotors, electric lamps and supplies should be kept in stock in sufficient quantities and convenient for immediate use.

Carefully selected men should be trained in the use of the helmets and resuscitating machines, and in the methods of fighting fires. The average man, unless specially trained, is useless in fighting fires. The foremen, shift bosses, electricians, pumpmen, watchmen and selected miners working in different places which are subject to fires should familiarize themselves with all things appertaining to the ventilation, and should know just what to do if fire occurs in their respective places. Rules to this effect should be printed, distributed, and understood by all concerned.

Watchmen should inspect the air outlets often and regularly in order to detect smoke as soon as possible.

Iron fire doors should be erected in connections near all timbered stopes, so that in case a fire gets beyond control the doors can be closed and the district quickly sealed up, in order to protect the remainder of

the mine. Similar doors should also be erected near timbered shafts, airways, etc., for protection against fires.

A fire signal should be in use at all mines and understood by all employees. A practice signal should be given occasionally. The following fire signal is in use here:

> "In case of fire, ring nine bells on the electric cage call signal. The engineer on receiving these bells will flash all electric light throughout the mine, nine times This flash will be repeated three times and followed by flashing the station signal on which level the fire is, three different times. Carmen and all others working where there are electric lights will notify those employed in stopes and other places where there are no lights. The trained firemen on the various levels will then take charge of the situation. Their first consideration will be for the safety of the men and then for the extinguishing of the fire and protection of property."

When a fire occurs in the fire district no signal is given, for the system there is such that the fire can be easily confined without danger from smoke or gas to the other parts of the mine.

When fire occurs in any other part of the mine, the signal is given and all men working where there is danger from suffocation are instructed to go to the surface or other places of safety. The next step is to get water on or over the fire as quickly as possible. When the fire is in the interior of the gob, it is sometimes necessary to drift from a chute or manway through the filling in order to get at it. Often, however, the fire can be extinguished by water from above. This is usually accomplished by driving holes in the filling with a pinch bar and turning water into these holes. For this work, hose nozzles made of 1-in. pipe with about a 3/8-in. hole are used. By this method, water can be scattered all over the stope. It will keep the fire from spreading and usually extinguish it.

In case a fire gets beyond control, the fire doors should be immediately closed, after which concrete bulkheads should be erected. It is sometimes possible to extinguish a fire in a sealed-up stope by water, through a diamond or churn drill hole. In sulphide stopes, however, it is usually necessary to burn the timber before the stope can be recovered. A fire, when apparently extinguished, may start again when the bulkheads are opened, unless the timber is consumed.

At the present time one large fire district here is being experimented with along these lines and with apparent success. Air in small amounts is gradually forced into the stopes to burn the timber. This causes the filling to settle, and, to avoid caving, more waste is put in from above.

Ventilation

Ventilation has received considerable attention here during the past 10 years, not only in the fire district but throughout the entire mine. Sulphide mines are naturally warm and disagreeable unless ventilated. Natural ventilation is used where possible; otherwise, fans are used to

blow the air where it is required. The ventilation of the fire stopes interferes with the remainder of the mine, since the air is taken from its natural source and forced into the fire district. Artificial ventilation is used for the middle and lower levels.

Fig. 3 is a plan of one level showing the location of the fan and the method of ventilation. All levels are connected to the air raise.

Fig. 4 is an ideal vertical section of the orebodies, showing the direction of the air currents, and the methods of ventilation.

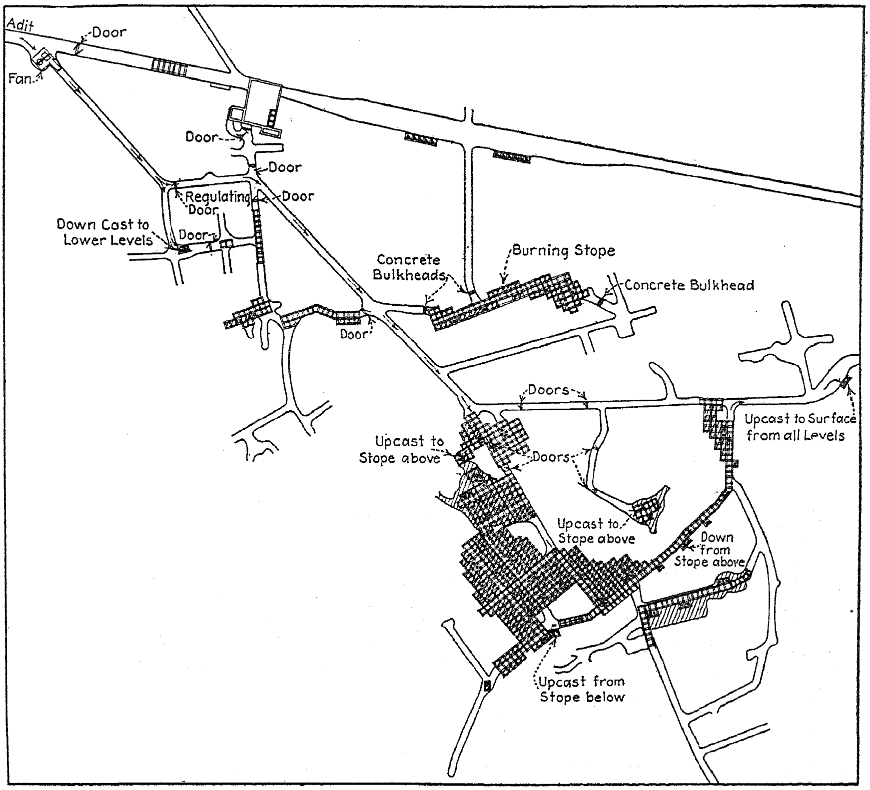

FIG. 3.—PLAN SHOWING LOCATION OF FAN AND THE METHOD OF VENTILATION.

Fans are installed in suitable places where they can draw air from the outside and force it into the different workings, thence to the air raises to the surface. The same airways and doors are used as for the fire district, except that no particular pressure is maintained, the air having as free an outlet as possible to the surface. The ventilation for any particular place or section is regulated and controlled by doors. For the proper ventilation of stopes, the air should have an inlet at one end and an outlet at the other. Most air is forced into the main working places and only a

sufficient amount is sent to the old workings to preserve the timber and make conditions unfavorable for fire.

The maximum temperature in the middle and lower levels is 80°F. It varies from 60 to 80° F. and averages about 72°, with small amounts of humidity.

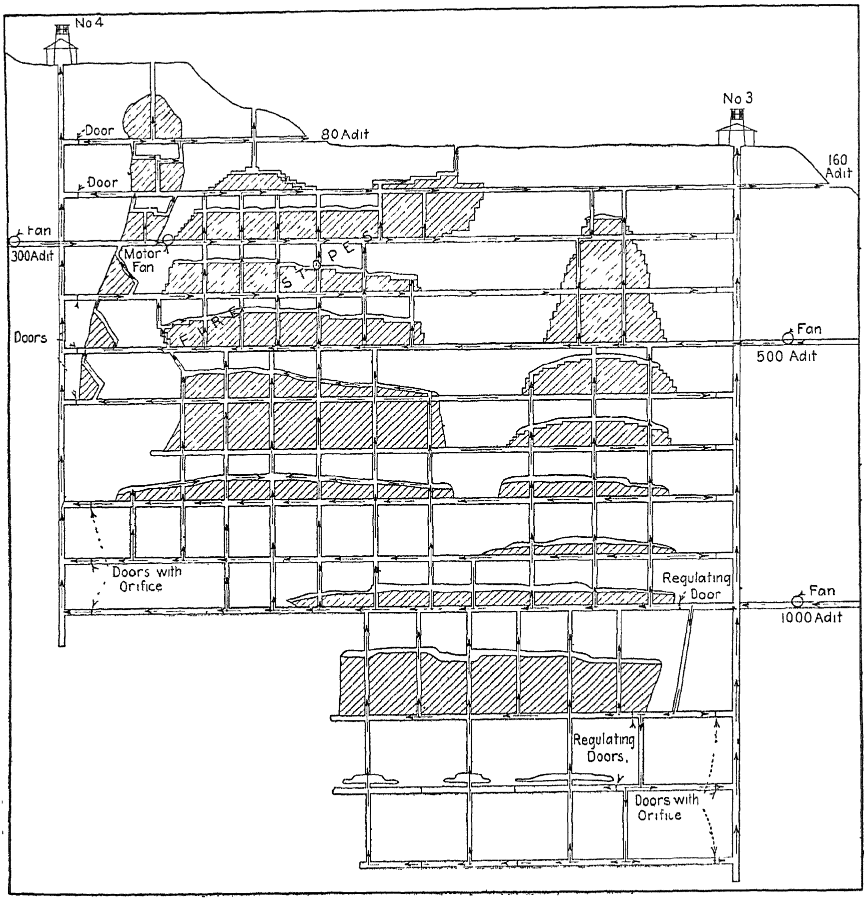

Fig. 4.—Vertical Section Showing Ventilating System of United Verde Mine, Jerome, Ariz.

Conclusions

For handling mine fires satisfactorily, the Plenum system as employed here has been a marked success, but the mining methods in connection therewith are necessarily expensive and inefficient.

Mine fires are the most disagreeable and dangerous conditions that are encountered in metal mining and every possible precaution should be exercised to prevent them. In addition to preventive measures, there

should be carefully trained firemen who thoroughly understand how to control and extinguish fires.

Discussion

Chauncey L. Berrien, Butte, Mont. (written discussion).—Having had much actual experience with mine fires which have occurred or have been active in the mines of the Anaconda Copper Mining Co. during the past 4 years and having lately visited the fire zone of the United Verde mine with Mr. Tally, I wish to congratulate him for the success which he has had in controlling this fire and making the mine perfectly safe for his miners. By this work he is not only keeping the property in operation but is reclaiming many lost orebodies.

The ventilation system as described by him is practically the same as that used under my direction at the Mountain View mine fire in Butte in 1913 and at the Pennsylvania fire in 1916. This same system is at present being followed at the Leonard mine to extinguish a fire which has been active for 10 years although bulkheaded.

The mining system in the Butte mines at the origination of these fires was the same as that at the United Verde, namely, square setting accompanied by the use of much timber bulkheading in the stopes owing to heavy ground. The pyritic content of the ore in Butte is not as high, but on the whole conditions have been the same, with the exception of actual location of the fire in the mine.

I agree with Mr. Tally that flooding, the use of CO_2 or any gas, and the use of steam are all impracticable unless the fire zone can be absolutely sealed. His system of upcast escapement shafts in the fire zone, the use of bulkheads and fans to keep the gases away from the firemen, must be resorted to. By the method he describes actual stoping may be carried on safely if started from the bottom of the fire and if the fire zone extends to the surface. In other words, the fire zone must be isolated from the rest of the mine by bulkheads and solid ground, with no ground above which can be damaged by spread of fire or caving.

The serious fires in Butte have occurred on or between levels in the mines where it was necessary to prevent their extension upward, downward and laterally while the work of actually extinguishing them was in progress. This necessitated proper bulkheading should we lose control of the fire, the installation of ventilation similar to Mr. Tally's and the systematic drilling of holes with diamond drills for distributing water on the fire and around it.

The Mountain View fire of 1913* was extinguished by the method described by Mr. Tally and the driving of laterals parallel to the fire

* See Fire-fighting Methods at the Mountain View Mine, Butte, Mont., *Trans.* vol. 52 (1915).

stopes. From these laterals many diamond-drill holes were drilled through which water was run onto the fire. This fire burned through stopes between the 200 and 600 levels.

The Pennsylvania mine fire of 1916 burned between the 1,000 and 1,300 levels and for a considerable lateral extent and was finally extinguished by the Plenum system and extensive diamond drilling. The diamond-drill work was in both cases the main factor. The details of this work were carried on by C. Edwin Nighman, at present Fire Superintendent of the Boston & Montana group of the Anaconda Copper Mining Co. and H. R. Tunnell, Foreman of the Pennsylvania mine, under the direction of the writer A description of this work will appear in the *Transactions* in the near future.

As I have said before, we are at present working on the Leonard-Minnie Healy fire with the prospect of regaining much ore, if not actually extinguishing the fire. The extent of the ground bulkheaded off from the rest of the mine is 600 by 400 ft. lying between the 600 and 1,300 levels.

This territory is filled with gases and an uncertain extent of smoldering fire. Our plan is to drive lateral drifts through the fire zone, keeping them in solid ground. From these laterals holes are being drilled with diamond drills into all stopes below, so that eventually we will have water entering all the stopes at 10-ft. intervals. We have been at this work about 1 year and have entered and reclaimed practically all of the 600 and 700 levels. We have also entered the 800 halfway and up to date the orebodies regained have warranted the installation of electric haulage on the 800. In doing this work we are using the Plenum system, and so far we have not had to resort to the use of Draeger helmets. While this work was really started as a matter of safety for the future of the mine, we are regaining orebodies at no greater expense than required for development of ore on new levels.

Our experience with fires in Butte has given us some thought as to their causes and led us to install all possible safeguards against their recurrence and to assure the necessary protection from new fires and the means to extinguish them.

The possible causes of mine fires are as numerous as of surface fires and one could set down a long list of them and the ordinary precautions for prevention. All of these should be known to the superintendent of any mine, and the subject is covered very thoroughly by George J. Young in "Fires in Metalliferous Mines."*

Over a period of about 10 years in 20 or more operating mines in Butte, the *actual* causes of mine fires have been: lighted candles left on or near timbers, short-circuiting and overheating of electric wires, switches, etc., due to carelessness in upkeep and operation, oily

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* *Trans.*, vol. 44 (1912).

waste, spontaneous combustion from foreign substances in gobs, and possibly incendiarism.

We realize that mine fires are possible even though the greatest precautions in the way of regulations, warnings, etc., are taken, and in the past 2 years we have installed every protection against the spread of fires, facilities for fighting them and means of rescue.

All of our main operating shafts are downcast, around the collar of each shaft we have 2-in. sprinkler lines; all water lines, air lines and electric cables are in downcast shafts, water-storage tanks of 3,000 gal. or more have been put in every 400 ft. along the shafts with pipe lines from them through main drifts, laterals and many stopes; water may be turned into all air lines also. All underground fan stations, transformer and switch rooms and electrical apparatus have been made fireproof for safe distances from them, overload and no-voltage release and auto starters are used on fans; all connections between mines are cut off by concrete bulkheads and iron doors except where stopes are continuous; a firebug (or watchman) goes through each shift boss' run in all mines after each shift; all electric switches are disconnected at shaft stations between shifts; telephones are used in all large mines; surface fans exhaust only; metal receptacles are used for oil waste and refuse and are in safe places; first-aid and Draeger helmet crews have been trained at each mine and hold positions where they can be notified of trouble immediately; signs denoting directions to other mines and main shafts are posted on all levels; an "out of the mine" danger signal is used; smoking in the mines is prohibited; metal sconces for candles are used entirely and failure to use them means discharge of man; mining methods are being adopted to eliminate as much timber as possible, and, finally, two fully equipped rescue stations on the surface are maintained.

At each of these rescue stations we have 30 Draeger oxygen helmets with accessories, lung motors, an automobile, a first-aid man on each 8-hr. shift, a smoke room for training and a man in general charge over all first-aid and rescue work. Classes are being held continuously and the number of capable firefighters is increasing. We also have a fire superintendent who acts with mine foremen in all underground fire work.

We have had six serious mine fires during the past 6 years and they have been extinguished or bulkheaded without loss of life except in the Pennsylvania mine fire of 1916. The loss of lives in this fire was due to the failure of the men in following directions given them for their safety. Work on all these fires extended over months. There have been about 12 other fires discovered and extinguished in a few minutes, or an hour at the most. This success has been due to preparedness, the good work of the helmet men, and extreme watchfulness. The expense of installation of any system or equipment for fire protection and fire fighting will be offset many times if such trouble arises.

C. W. Goodale, Butte, Mont.—I wish to say that I have before me a discussion of the fire that occurred in the Pennsylvania mine in Butte, by Messrs. C. E. Nighman and R. S. Foster, but it is quite long and the conclusions are very much the same as those given by Mr. Berrien in the discussion which has just been read. I will say briefly that the Pennsylvania fire started Feb. 14, 1916, about 8:45 p.m., at or very near the 1,200 air-shaft station. This fire resulted in the loss of 21 lives; however, had these men followed the instructions of those sent to warn them, a maximum of nine, and probably only seven, fatalities would have occurred. At the time the fire occurred the main working shaft was upcast, and while this condition caused no loss of life, considerable discomfort resulted from the presence of smoke and gases, and rescue work had to be abandoned until this shaft was made downcast, which was accomplished by placing a suction fan over the air shaft. It should be stated that when the fire occurred preparations were under way to make the working shaft a downcast.

It is extremely important that working shafts be downcast wherever possible, for the following reasons: (*a*) In case of fire, ordinarily men can come to the shaft and be hoisted without danger of suffocation (in some cases action of fire has been known to change the direction of air in shafts); (*b*) Where cold air is coming down a shaft, the tendency of the ground to slough is greatly lessened; also, timbers last much longer when not exposed to hot, humid air coming out of the mine, and as working shafts must bear the strain of traveling skips and cages it necessarily follows that they should be favored where natural conditions permit. Furthermore, air shafts can usually be retimbered without curtailing production.

On the other hand, during the winter months in cold climates, ice collects in downcast shafts even where there is very little moisture. It is necessary to clear the ice by chopping it out, which is a disagreeable and somewhat dangerous task. Furthermore, it is hazardous and uncomfortable to do repairing in downcast shafts during cold weather.

At the present time the working shafts of the Anaconda Copper Mining Co. in Butte are downcast wherever possible.

The Chairman (Percy G. Beckett, Globe, Ariz.).—I think, as Mr. Tally said and as Mr. Goodale has brought out, fires are exceedingly disagreeable and dangerous things to have in a mine; and I think it is a vital subject for us all. There are several big mining districts having fires burning in those districts for a long time. It is one problem to keep the fires under control, and another problem full of importance to us all to prevent fires from starting from the numerous sources from which they can start.

Gerald Sherman, Bisbee, Ariz.—We have been unfortunate enough to have some fires, and we have used very much the same methods that

have been described by Mr. Tally and Mr. Berrien. I am not sure that the members here are familiar with Mr. Berrien's paper of last year; but I think there is a distinction between what Mr. Tally is trying to do or doing and what Mr. Berrien is doing. From the shape of the orebodies in the United Verde, it is very much more difficult to put them out than in Butte. It is very difficult to get at them. The ground over the fires is soft, crushed and open; and I doubt if drill holes could be put in to get water on the fires. I think Mr. Tally's effort, although he has put out many fires, has been principally toward controlling them by the Plenum system. He recovers the ground by blowing the fire away from the working places and driving the gases to the surface through broken ground or some passage which does not interfere with other work. This does not put the fire out unless it is kept away from other timbered ground until it goes out of itself. The only way I know by which fires have been put out in this country is by putting water on them. Mr. Tally has put fires out by that method, and Mr. Berrien has also. In the cases at Butte, most of the veins were in comparatively hard rock. They are not vertical, as I remember it, but very steep. By drilling from the hanging wall, which is safe to work in, you can pour water on the fire successfully, and that is the way we did it. In one of our fires, we were lucky enough to get water on top of the fire in such a way as to drown it, and it was finally put out so we could get in the ground and prove it. Another was kept down by putting water on it until we were able to get to the fire area and by following the smoke to dig it out, but that was a small fire. In one of our fires, which has been burning for some years, and which is now isolated, we tried to drill holes over the top with diamond drills, but it was unsuccessful, as the ground was too much broken. We were never able to get near the point we were driving for. I think the precautions which Mr. Berrien has suggested are very well worth carrying out. All of our fires have been spontaneous, as far as we know, and they have all occurred, with one exception which is rather doubtful, in the centers of filled and abandoned stopes, which were inaccessible, and we were not able to reach them quickly. We had to isolate them and cut them off from active workings and keep them from spreading. I think they are undoubtedly caused by the heating of sulphides in stopes where they are loose and granular, and where there is little ventilation. I believe if raises could be kept open for free circulation of air, the heat would be carried off so that the temperature would not rise to the point of ignition, but I am not sure of it. In some of our shafts, we are laying pipes and sprays to water the shaft in case of fire, and also putting up doors to cut them off and stop circulation from the shaft, but we do not care to say very much for the present.

J. P. Hodgson, Bisbee, Ariz.—Mr. Chairman, from my experience in mining, covering a period of about 30 years, I would say that I think

the most prolific source of mine fires is the use of candles, owing to the carelessness of the men in leaving what we call "snuffs" around their working places. I know of at least two mine fires in Michigan where I was operating that occurred in this manner. One of these we put out with water; in the other we sealed the shafts and shut down the mine for 3 weeks before we extinguished the fire. Regarding our experience in copper mining, as Mr. Sherman has said, we have had a lot of experience in fighting fires, but we are not yet sure as to which is the better method. One thing I might mention in this connection that might be of assistance to others is that we keep on hand a 15-hp. electrically driven blower mounted on a truck, together with 1,000 ft. of wire for connections and 500 ft. of galvanized 10-in. blower pipe and a roll of brattice canvas. This equipment is kept in a certain place. When a fire alarm is turned in this equipment can be sent underground and put in working order in from 30 to 45 min. With this equipment we can force back the smoke in the drifts and sometimes succeed in reaching the fire, and in putting it out before very much damage is done. We have a fire in the Lowell mine that has been burning for over 6 years and we have gradually encroached on the fire area by putting in concrete bulkheads and thus getting ore that could not be mined before on account of the fire. We are at the present time on the 800-ft. level of the Gardner mine driving a series of drifts over a fire area similar to those explained in Mr. Berrien's paper. We are putting water in these drifts so that it may percolate through the broken ground to the fire zone below. This fire is in an old sulphide area which has been stoped out, and I believe in a year or two we will succeed in extinguishing this fire.

The Chairman.—Is there any more discussion on this paper? I would like to ask Captain Hodgson from his experience fighting fires at Bisbee what he considers the main fundamental precautionary methods that should be taken in the average mine. Is it more a question of shafts in the right place, upcasts and downcasts, or water lines laid throughout the mine, or a question of blocking the mine off into sections that can be closed quickly, or what?

J. P. Hodgson.—I believe that all of the things you have mentioned are very good, but in such mines as the Copper Queen it is almost impossible to do all of them in every case as the property is so extensive and as a usual thing the fire breaks out where you don't expect it. After careful investigation we adopted this blower and have it in reserve so that we can get quick action after an alarm is turned in. It is lowered and in operation in from 30 to 45 min. in any part of the mine, and we believe it is very good for our conditions. Also, at the Copper Queen, we have fire doors to block off the different parts of the mine so that in case of a serious fire in one part it can be cut off and the balance of operations

proceed, but I fear I could not say which is the best way. All the things mentioned are good but the fire usually starts in the wrong place and without giving due notice. We also believe that our method of ventilation, keeping the mine under pressure, materially assists us in keeping back our fires.

C. A. MITKE, Bisbee, Ariz. (communication to the Secretary*).—I am interested in Mr. Tally's methods of fighting mine fires at the United Verde Copper Co. because he has developed a system of mine ventilation which has made possible the mining of orebodies that are adjacent to fire districts, as well as orebodies which are actually on fire. Of course, when preliminary work is being done in the way of development of these fire areas there is considerable heat to contend with; but after the ventilation has been arranged the miners are able to stope out the ore with almost the same comfort as in ordinary orebodies which are not on fire.

Gardner Fire District.—Since the pressure system of ventilation was successful at the United Verde Copper Co., it was adopted at the Copper Queen.

There are similar mine fires in the Copper Queen, as, for example, the 9-1 district of the Gardner division where a fire has been known to exist for some time. About 2 years ago a large amount of carbon dioxide came from this district on the 1,000 level, in which three men were overcome. Steps were taken to inclose it by means of doors, and to have air pressure of about 7 lb. per square foot surrounding the district to prevent its spreading. A raise was driven between the 800 and 900 levels and intermediate drifts separating the fire country from live workings. On the 800 a drift and a number of crosscuts were driven over the hot area. Water was run into these intermediate drifts and crosscuts so as to cool the country rock. An exhaust fan was used to draw the air from the top of the drifts and allow fresh air to come into the headings. At present the amount of gas and heat coming from this district has been decreased and it is believed that in a short time the district will be cool enough so that the usual ventilating system will continue to cool it without the continued use of water.

Cananea Consolidated Copper Co.—About 2 years ago there were several fires in the Veta Grande and Oversight mines in Cananea. These fires were located principally in timbered drifts and stopes. While some of them were easily extinguished, nevertheless one of them in the Veta Grande required considerable work done by men using oxygen helmets. The greatest difficulty was experienced in getting the fire under control, on account of there being no comprehensive scheme of ventilation.

Rules.—There should be a set of rules at every large mine which

* Received September, 1916.

should prescribe first for the safety of the men and then for combating the fire. The importance of this may be illustrated by the Holbrook mine fire which occurred 3 years ago. As two men were taken out of a raise on the 500 level when they were overcome with gas, we asked the foreman if he had a pulmotor. He said "it is here in this drift." The pulmotor had been there for 20 min. and it was used effectively on a shift boss and assistant superintendent. When helmets were required, they were available immediately. The same was true with canvas, axes, saws, etc. During the night the fire was sealed up entirely and the mine was in operation the next day. The importance, therefore, of having all necessary supplies on hand in case of fire cannot be overestimated.

Cost and Extraction in the Selection of a Mining Method

BY C. E. ARNOLD, B. S., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

IN attacking the problems of mining and treating large disseminated copper orebodies such as those occurring in the Miami or the Ray district of Arizona, one of the vital questions to be decided is, "What is the value of the orebody per ton in place?" The importance of this question lies in the fact that its answer helps to reach a decision concerning the method of mining to be employed.

Let it be assumed for the moment that milling and smelting practice, especially the latter, is more or less standardized, and has a cost incapable of much variation, while the cost of mining has a considerable range owing to the large number of mining methods by which a mine may be exploited. For instance, a method may be adopted whereby all the available ore can be extracted perfectly clean (*i.e.*, unmixed with barren material) at a relatively high cost; or a method may be used involving the necessary dilution of a portion of the ore with waste, in order to recover that portion, thus giving a relatively poorer extraction but nevertheless a lower working cost than the former method. Naturally, between these two extremes there lie intermediate methods.

In order to arrive at a solution of a mining method problem that is of any value, it is necessary that accurate and complete estimates of mining, milling, and smelting performances and costs shall be established. By way of illustration is given a case of mining, by the system of undercutting and caving, a large orebody with an average content of 1.50 per cent. copper supposedly all in sulphide form, to which case has been applied those results in mining operations which one may fairly assume are to be expected when viewed in the light of G. R. Lehman's ore-drawing experiments, which form the subject of a paper to be presented at this meeting.

It is assumed that the total costs will be as shown in the table on page 204.

To attain these costs, a daily production and treatment of 14,400 tons has been assumed. Other assumptions are as follows: 15 per cent. of orebody is to be extracted clean by development work and undercutting; 63.8 per cent. (75 per cent. of 85 per cent.) is to be extracted clean by caving, leaving 21.2 per cent. part of which is to be extracted by diluting with waste, and the remainder lost. Dilution and extraction are to

Costs on 1 Ton of Clean Ore

Mining, including prepaid development	$0.600
Coarse crushing.	0.024
Freight on ore...	0.015
Concentrating.....	0.462
Freight on concentrate.....	0 002
Smelting, freight on copper, refining, marketing, etc.	0.737
Amortization..	0.120
Total.........	$1.960

be related as shown by the curve in Fig. 1. The copper tenor of the concentrate is to vary uniformly between 25.33 per cent. and 30.00 per cent. on 0.80 per cent. and 1.50 per cent. ore respectively. Mill extraction is to vary uniformly between 69 per cent. and 83 per cent. on 0.80 per cent. and 1.50 per cent. ore respectively. The selling price of copper is to average 14 c. per pound. The life of property will be 12 years. Smelting is to be performed under contract with an outside company.

In the above set of assumed costs the charge of $0.12 per ton of ore for amortization represents the outlay necessary to redeem in 12 years

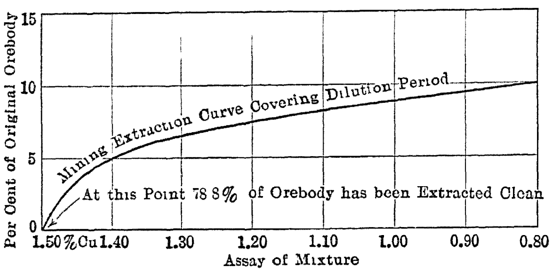

FIG. 1.—SHOWING EFFECT OF DILUTING THE LAST DRAWN ORE WITH CAPPING UPON COPPER ASSAY OF THE MIXTURE

all expenditures for plant (not including smelter), lands, railroad, water supply, etc., other than the expenditures required in preparing the mine for the assumed daily production. This latter expense is covered by a charge of $0.20 per ton, which is included in the estimate of $0.60 per ton for mining. The amortization fund is considered as an annuity invested at 4 per cent.

It should be borne in mind that as long as the ore is undiluted with waste, the above total cost of $1.96 is all that is applied to the ton in place, and that the instant dilution starts the cost per ton in place increases because the added waste, until it disappears in the mill tailing, goes through all the operations to which the ore is subjected. Furthermore, when the ore is diluted, the mill extraction, and consequently the smelter return, decreases, likewise the copper tenor of the concentrate decreases. These disadvantages, however, are to a certain extent offset

by the fact that on dilution, the increasing concentration ratio and the decreasing copper content of the concentrate lower the smelting cost on the ton of ore in place.

Taking into account all these variables, the curves shown in Fig. 2 were plotted, and then by using these in conjunction with the curve of Fig. 1, the curve of Fig. 3 was established, showing the extent to which

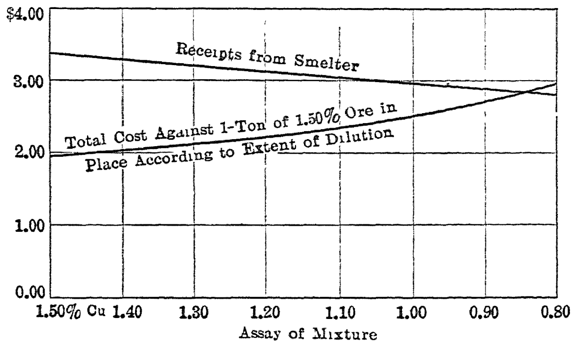

FIG. 2.—SHOWING DECREASING SMELTER RECEIPTS AND RISING COSTS WITH INCREASED DILUTION.

dilution could be carried to secure the maximum profit from that ore remaining in place after the extraction of all available clean ore.

The conclusions resulting from this set of calculations are as follows:

1. To obtain a maximum profit the average assay of the dilution product will be 1.28 per cent. copper, the end point being 0.845 per cent., as shown in Fig. 3, at which point receipts from sales of copper just balance total costs.

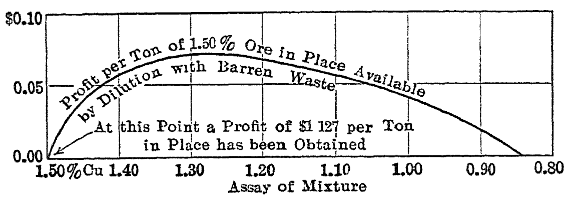

FIG 3.—SHOWING MAXIMUM POSSIBLE PROFIT WITH ORE DILUTION INCREASED UNTIL ORE ASSAY IS REDUCED TO 1.28 PER CENT. OF COPPER.

2. The total ore recovered will be 85.5 per cent., as shown in Fig. 1, of which 78.8 per cent. will be extracted clean and 6.7 per cent. will be extracted in conjunction with 1.15 per cent. of waste.

3. From 100 tons of ore in place there should be recovered:

78.8 tons clean, at a profit of $1.430...	$112.68
6.7 tons diluted, at a profit of 1.063...	7.12
85.5 tons at a total profit of...	$119.80

Expressed otherwise, the above might be stated thus:

78.8 tons clean ore at a profit of $1.430	$112.68
6.7 tons clean ore at a profit of 1.273	8.53
Gross profit	$121 21
1.152 tons waste at a loss of 1.221	1.41
Net profit	$119 80

This is equivalent to $1.20 per ton in place.

To duplicate this performance by the employment of a mining system that would give, say a 95 per cent. extraction of clean ore (this representing the end of mining operations), a mining cost increased from $0.60 up to $0.77 per ton would have to be borne as indicated below:

	Per Ton
Receipts from smelter	$3 39
Costs other than mining	1 36
Difference	2 03
Cost of mining.	0 77
Difference	1 26
Mining extraction, per cent	95
Profit per ton in place	$1 20

Further, it may be stated that on some particular ore, of higher copper content than 1.50 per cent., it would be equally profitable to apply either a low-cost, low-extraction mining method, or a high-cost, high-extraction method; and with further increase in the copper content of the ore the high-cost, high-extraction method would yield the greatest profit per ton in place.

By the application of similar reasoning to various mining problems such as that outlined above, it will be found that the resulting figures indicate plainly the mining method that will yield the greatest ultimate profit. Should the calculations shaow tht there is little difference in the results of several methods, the logical decision would be to use that method in which the element of doubt is smallest regarding the realization of the estimated costs.

Discussion

The Chairman (Percy G. Beckett, Globe, Ariz.).—When you are at Miami tomorrow, you will see a very interesting comparison on the surface of the two mining methods that are being employed there largely. In the case of the surface over the Inspiration mining method, you won't see very much left of the surface. Everything has gone down. In the case right at the line of the Inspiration ground and the Miami, you will see that the surface over the top slicing which the Miami company has been doing in that countıy has been riding down evenly and uniformly; and it gives a very marked comparison of the two

types of mining, and, to a certain extent, of the different degrees of waste dilution. I would like to hear from Mr. Maclennan on this subject.

F. W. MACLENNAN, Miami, Ariz.—A paper based on percentage extraction by shrinkage stoping is a difficult one to discuss for the reason that it is hard to arrive at extraction figures until a section has been completely mined out. In the Miami mine we are using a method of shrinkage stoping with narrow stopes and pillars. This ore has been completely broken up and we have started in at the far end to draw off the ore mass maintaining the top of the broken ore dipping at an angle of 60° away from the direction in which we are retreating. A very important point, I think, about drawing a large mass of ore down at an angle rather than trying to maintain the top of the ore more or less horizontal, is that drawing at a steep angle has a tendency to create a horizontal movement in the ore mass as well as the vertical movement of the ore traveling down to the drawing off chutes. We know that this takes place on account of the horizontal movement of marked blocks which were placed in the broken ore mass at the time of stoping. This horizontal movement tends to cut off and break up pipes and funnels which are liable to open up through the ore from the drawing off chutes, allowing waste to drop down to the chutes before the broken ore surrounding the open pipe has been drawn in.

I noticed one remark made by Mr. Lehman in presenting his paper, to the effect that ore is drawn off from the finger raises until capping appears in these raises. When this occurs the drawing off is considered completed and these raises are sealed. Our experience in drawing off ore in the Miami mine would indicate that this is not always reliable. In many cases we have had chutes piped clear through to the surface, allowing capping to drop down to the chute when only 10 or 15 per cent. of the expected tonnage from this chute had been drawn. When this happens we seal the chute off temporarily and draw other chutes surrounding it for a few days and then resume drawing in the chute in which the capping appeared with the result that, usually, after drawing 15 or 20 tons of capping the ore starts to run again and may continue to run for several hundred tons of clean ore before capping again appears. For this reason I believe it is very important to establish a horizontal movement in the broken ore mass in order to break up these vertical pipes to the capping and also to help break up the thin pillars which were left between the stopes; and I also believe it is important to have the tonnage record from each individual chute so that an indication is had of whether capping appearing in a chute is only temporary or whether it marks the final stage of ore drawing. I think both of these points have a very important effect on the final extraction and dilution of the ore drawn.

Power Plant of the Burro Mountain Copper Co.

BY CHARLES LEGRAND,* DOUGLAS, ARIZ.

(Arizona Meeting, September, 1916)

THE power plant of the Burro Mountain Copper Co. is located near Tyrone, N. M., at 5,950 ft. elevation. It is interesting because it uses the largest stationary Diesel engines in the United States.

The general layout of the plant is shown in Figs. 1 and 2. The building is of steel-frame construction with hollow-tile walls and corrugated-iron roof; the floor is concrete and all windows have steel frames, making a fireproof building.

The building is made to accommodate one more electric generator unit, which is being installed; also two 4,000-cu. ft. air compressors direct-connected to three-cylinder engines with the same size cylinders as the electric units.

The power plant consists of two 815-kva., 60-cycle, three-phase, 6,600-volt, 180-r.p.m. generators, direct-connected to five-cylinder, two-cycle Diesel engines; the necessary circulating pumps for cooling water, cooling towers and oil storage tanks.

A portion of the power is used in the same building as the generating units to run one of two 200-kw. rotary converters delivering 260-volt direct current, and a 2,500-cu. ft. air compressor, delivering air at 95 lb. pressure, direct-connected to a 400-hp., 6,600-volt, three-phase synchronous motor.

The remainder of the power is used to run a 1,500-ton concentrating mill 3½ miles from the power house, various hoists, pumps and motors about the mines and the lighting and water-works of the town of Tyrone.

The direct current is used for electric traction in the mine, also to run hoists and motors that were installed before the new power plant was built.

Due to the large hoists that take starting peaks of 250 kw. and the 10-ton electric locomotive that takes at times 150 kw., the load is extremely variable, as shown in Fig. 3.

The two five-cylinder vertical Diesel engines, which are rated at 1,250 b.hp. at sea level, have cylinders of 525-mm. diameter (20.6 in.) and

* Consulting Engineer, Phelps, Dodge & Co.

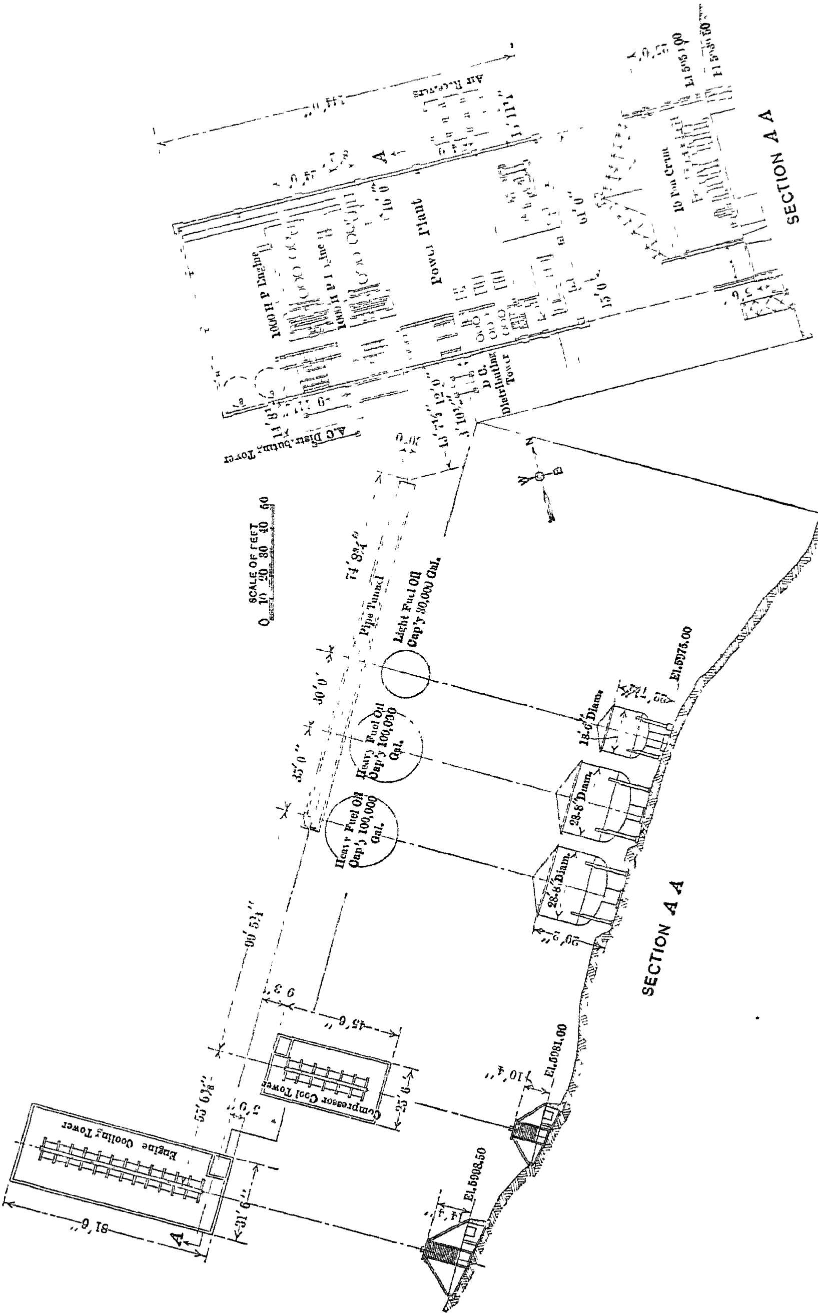

FIG. 1.—GENERAL ARRANGEMENT OF POWER PLANT EQUIPMENT, BURRO MT. COPPER CO., TYRONE, N. M.

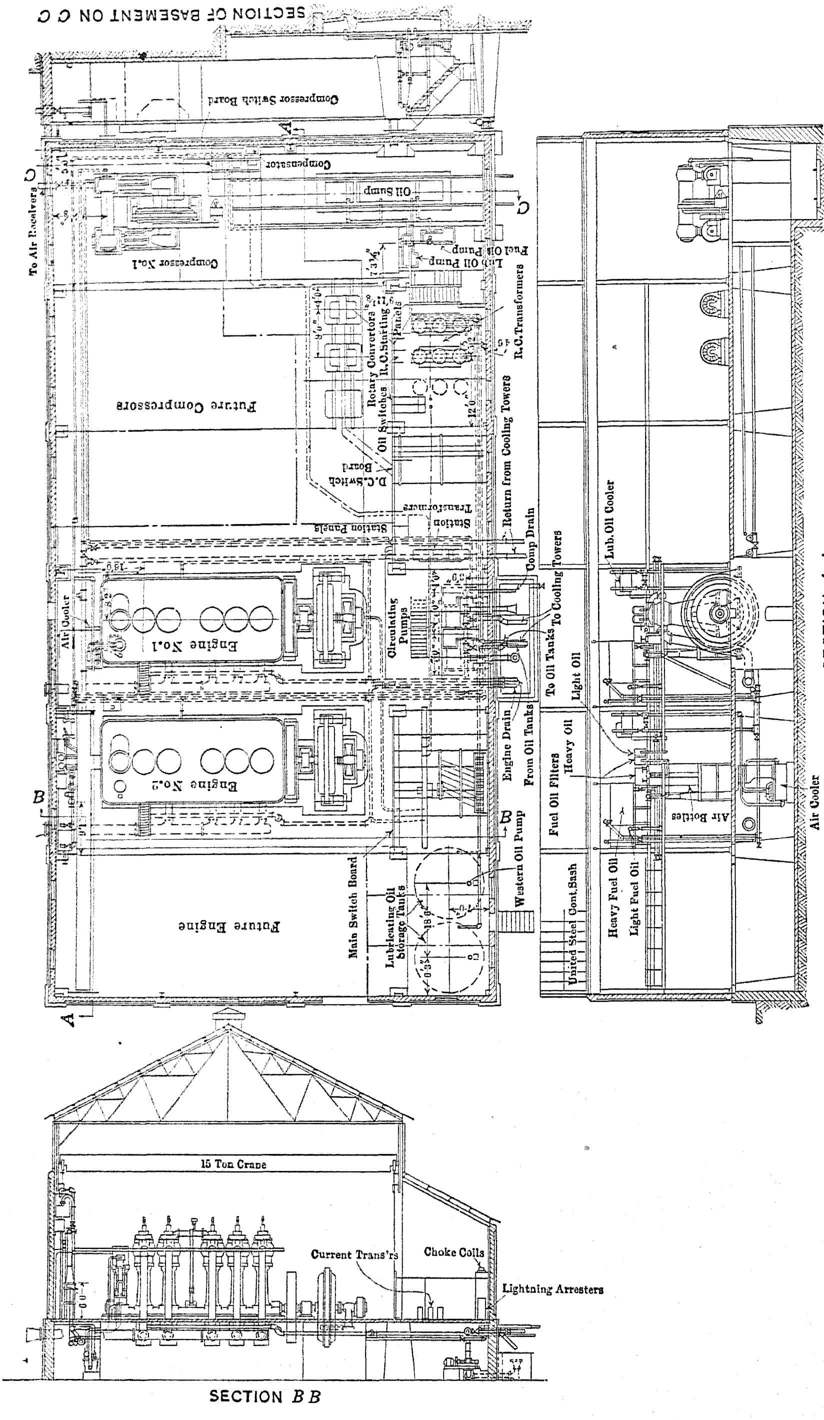

FIG. 2.—GENERAL ARRANGEMENT OF POWER PLANT OF BURRO MT. COPPER CO.

660-mm. (26 in.) stroke; each engine has a scavenging cylinder of 1,050-mm. (41.25 in.) diameter and 600-mm. (23.6 in.) stroke; also a three-stage, four-cylinder high-pressure vertical compressor, both directly connected to the engine. This compressor delivers the air necessary for fuel injection and for starting the engine. The scavenging pump, which is larger than usual, delivers the air to blow off the products of combustion and fill the cylinders with fresh air at the beginning of the stroke This pump was increased in size to be able to fill the cylinders with air at 2½ lb. gage pressure at the beginning of the stroke. This gives nearly the same initial absolute pressure and allows the engine to generate nearly the same indicated horsepower as it would at sea level. The work done in the scavenging pump is, however, increased and the horsepower available is approximately 95 per cent. of sea-level output.

The fuel consumption per horsepower is increased over sea-level conditions because of the extra work of the scavenging pump and the resulting lower mechanical efficiency of the engine when operating at the elevation of this plant.

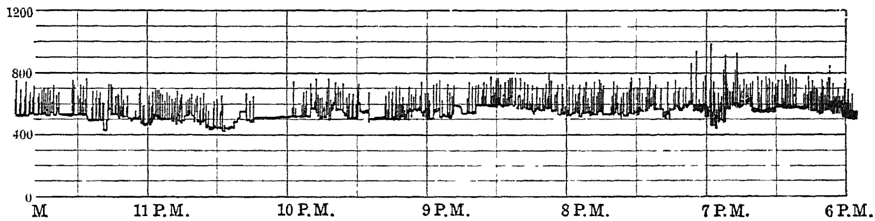

FIG. 3.—SECTION OF CHART SHOWING VARIATION IN POWER LOAD.

The plant has been in operation for 17 months, but as the mill was not ready until the middle of April, 1916, up to that time the load was carried on one engine.

The fuel used is California asphaltum base oil 14° to 18° Bé. gravity, averaging 16° Bé. and 18,360 B.t.u., costing $1.85 to $1.98 per barrel. The number of heat units in the oil was not determined at Tyrone, but at another plant which gets the oil from the same shipping point and under the same contract.

The pistons, cylinders, heads and exhaust pipes are water-jacketed; the jacket water of the exhaust pipes, which can be easily varied in temperature without danger, is used to heat the fuel oil to 120° F. as the heavy fuel oil cannot be used if cold. The engines are started with a lighter fuel oil of about 25° Bé. and run on this oil until the heavy oil is heated to the required temperature; when shutting off an engine, it is run for a few minutes on light oil to fill up the oil piping with light oil.

The curve in Fig. 4 shows the oil consumption per kilowatt-hour delivered at the switchboard when engines were tested new; it is the aver-

age of several tests. It is interesting to compare the test figures with the actual consumption as shown on power reports of the company. Although individual months show variations from the curve, the yearly average for 1915 checks very closely with it, showing that the efficiency of the engines is well maintained and that the variable load has very little detrimental effect on the fuel efficiency.

By the courtesy of Phelps, Dodge & Co., owners of the Burro Mountain Copper Co., I am allowed to publish in the accompanying table the operating costs from Jan. 1, 1915, to June 1, 1916. These costs are as they stand on the books and are higher than is to be expected. They cover the running cost of the complete plant, exclusive of the air compressor. These costs do not include taxes, overhead charges or deprecia-

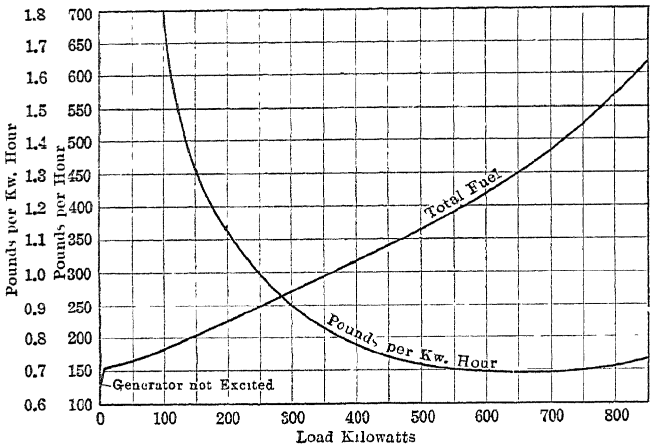

FIG. 4.—AVERAGE FUEL CONSUMPTION TESTS ON DIESEL ENGINE, BURRO MT. COPPER CO.

Diesel Engine: 5 cylinder, 1250 hp. (sea-level rating), 2 cy. ch.
Plant Elevation: 5,950 ft.
Fuel: 16° Bé California crude oil, 18,360 B. t. u. per pound.

tion. The cost per kilowatt-hour available is based on total power generated less that used for auxiliaries in power plant, such as lighting, circulating pumps for oil and water, etc. To simplify bookkeeping, the compressor is not charged with the cost of circulating its water or with its proportion of the lighting.

The labor cost is high because we have operated from the first with the full labor needed for operation when the power house is delivering the power necessary to run the mines and mill at the rate of 1,500 tons per day, a condition that has not yet been reached.

The repair cost is high, for soon after operations were started the construction account was closed and several items, such as labor for installing recording thermometers, measuring gages, some lubricating-

TABLE 1.—*Burro Mountain Copper Co., Cost of Power*

1915	Labor	Fuel Oil	Lubricating Oil	Miscellaneous Supplies	Repairs	Total	Total, Kw.-hr.	Available Kw.-hr.	Cost per Kw.-hr. Available	Pounds Fuel Oil per Kw.-hr.	Load Factor	Average Load
Jan..........	$602.15	$754.79	$98 50	$34.50	$63.93	$1,553 87	101,315	93,322	$0.01665	1.29	0.18	136
Feb..	796.60	742.50	72 60	33.31	198.44	1,843.45	77,400	69,512	0.02683	1.68	0.15	115
Mar..............	782.72	859.50	80.06	50.71	179.18	1,952.71	143,820	135,025	0.01446	1.02	0.26	192
Apr...... .	731.83	919.71	71.45	85.67	291.32	2,099.98	134,473	125,795	0.01669	1.13	0.25	187
May..............	740.86	959.00	128.86	28.26	456.64	2,313.62	130,690	122,703	0.01880	1.24	0.24	176
June	705.83	882.92	142.96	29.47	393.57	2,154.75	152,646	145,347	0.01482	1.00	0.28	212
July..............	728.01	1,043.44	111.90	51.56	443.64	2,378 55	159,742	151,089	0.01574	1.16	0.30	228
Aug..........	762.68	1,037.96	110.99	34 08	477.01	2,422 72	155,800	149,397	0.01622	1.14	0 28	209
Sept....	673.05	1,046.87	122.77	15.76	507.69	2,366.14	185,753	177,853	0.01330	1.00	0 34	253
Oct.	705.01	1,284.25	122.66	17.97	689.58	2,819.47	275,836	267,174	0 01055	0.81	0.49	370
Nov..	692.50	1,393.87	128.69	40.31	363.06	2,618.43	288,394	280,323	0.00934	0.84	0.53	400
Dec..............	702.45	1,492.40	132.50	9.81	316.58	2,653.74	335,702	326,580	0.00813	0.77	0.60	450
... ...	8,623 69	12,417 21	1,324.48	431.41	4,380.64	27,177.43	2,141,571	2,044,125	0 01330	1.00	0.33	214
1916												
Jan	$712.15	$1,527.09	$124.88	$30.60	$285.79	$2,680 51	349,154	339,822	0 0078880	0 764	0 63	470
Feb. .	694 43	1,671.94	127.57	21.66	264.84	2,780 44	353,119	344,553	0 0080697	0.750	0.70	520
Mar . . .	791 45	1,399.03	135.93	20 60	497 61	2,844.62	370,790	362,149	0 0078518	0.750	0.68	505
Apr. . .	768.65	1,897.18	174.38	21.07	574.02	3,435.30	451,590	441,557	0.0077799	0.780	0.58	433
May. . . .	961 97	2,522.00	186 66	29 14	1,024 58	4,724.35	559,550	546,282	0.0086635	0 837	0 57	425
	3,928.65	9,017.24	749.42	123.07	2,646.84	16,465.22	2,084,203	2,034,363	0.0080935	0.780	0 62	464

oil piping, etc., were charged to repairs. During the first year we also took apart the pistons, valves, etc., of the engine to satisfy ourselves that everything was going right and to determine if possible at what intervals this work would have to be done. This is shown by the relation of repair labor to material used in the repairs. The third engine is being furnished with scavenging valves having removable seats, and we decided to alter the valves of the two engines already erected while we needed only one engine in service.

The cost of repairs is shown in Table 2.

The repairs to circulating pumps and water piping are repairs outside of the engine water piping; the repairs on lubricating-oil system are also the repairs outside of the engine. The item of other repairs covers all repairs to building, switchboard, lighting, fuel-oil system and rotary converters. The latter should not be included under the cost of alternating-current power, but to separate them would have involved a great deal of work and the final result would not be appreciably different.

TABLE 2.—*Burro Mountain Copper Co. Power Plant Repairs*

1915	Engines		Circulating Pumps and Water System		Lubricating-Oil System		Other Repairs	
	Labor	Material	Labor	Material	Labor	Material	Labor	Material
Jan.	$46 40	$5.21	$1 50	$2.82	1.10	....	6.90	
Feb	142 95	2.61	36 10	2 49	. .	...	11.70	2 59
Mar	144.30	7 67	4 60	....	..	0.78	18 85	2.98
April	171.05	35.46	2 70	14.08	0 35	0.48	48.25	18.95
May	366.60	6 80	26.70	7 85	5.15	...	42.30	1.24
June	297.50	3.52	53.80	20.71	2.58	2.58	14.50	0 96
July	234.60	3.51	79.60	58.56	.....	....	50.10	17.27
Aug.	303.55	70.65	63.80	6 56	12 00	1.10	12.95	6.40
Sept.	223.75	7.67	70.30	40.47	33 95	11.79	95.95	23.81
Oct.	280.80	261.37	61 15	16 43	16.85	6.15	33.30	13.53
Nov.	299.67	5.92	7.95	3 04	2.75	0.15	18.70	24.88
Dec.	229.75	51 85	21.00	0.05	0.95	0.43	8 80	3.75
Jan.	169.00	0.34	19.70	0.10	59.45	9.54	12 20	15.46
Feb	167.75	3.68	17.35	2.39	14.25	0.44	21.40	37.58
Mar.	284.95	152.50	....	.....	9.40	2.56	36.05	12 15
Apr.	74.30	3.33	.....	...	142 00	120.43	122 55	111.41
May.	320 95	483.38	8 15	0.47	80.70	29.45	52 80	48.68

The water used in cooling the engines is mine water, which apparently has no bad effect on cast iron, but has a strong action on steel, so that all of our oil filters had to be rebuilt.

The circulating pumps come in for a large repair item due to breakage of check valves above the pumps which fell in and wrecked the pumps.

The two largest items of repairs on the engines were, one main bearing

that was drained of its oil by a mistake of the oiler the first night we operated and one high-pressure air-compressor head that was wrecked by starting the engine without draining the air cylinder, after it had been idle some weeks. A small water leak had filled the cylinder with water and the head broke at the first revolution of the engine.

The greatest trouble we have had, not due to carelessness, is with the helical gears driving the camshaft. These gears have a tendency to get loose and one of them split from the keyway. We have not ascertained the exact cause of this trouble, which is now much reduced since iron bands were shrunk on the hub of the gears.

Two important items of running expense were doubtful when the engines were purchased: One was lubricating oil and the other the maintenance cost. The first one is now settled and we hope to reduce the cost from that shown in the table by the use of separate pumps to each point where the oil is delivered in the cylinders, insuring a definite quantity of oil. The present engines have one pump delivering to four points and it happened once that one of the oil pipes became plugged and gave slight trouble on one piston.

The maintenance is still in doubt, and cannot be determined until we reach and maintain regular operations for some time; but the writer thinks it will show a decrease from the figures shown.

The water-cooling connections to the pistons were expected to give some disagreeable troubles, but we have had practically none.

The operation of the engines in parallel is very satisfactory even when the load is light. In actual operation, with one engine in service, the peak load carried has been higher than was expected to be possible. Under test, 900 kw. was the limit with water-rheostat load, but in service, peaks up to 1,000 kw. have been carried for several seconds without seriously slowing down the engine.

On the whole, the operation of the plant has been satisfactory and the costs obtained are considerably better than with a steam plant of the same capacity run under the same conditions, amply justifying the extra investment for this type of engine where the cost of fuel oil is high.

Discussion

The Chairman (B. B. Gottsberger, Miami, Ariz.).—I think this experience of the Burro Mountain Copper Co. with oil engines as described by Mr. Legrand comes at a very opportune time. The fuel almost universally used in the mine power plants of the Southwest is oil. Additional economy in its use for power production is much to be desired. In the case of the Miami Copper Co., we have lately found it necessary to enlarge our power plant and, based on the experience of the Burro Mountain Copper Co., we have decided to install oil engines similar to those in their plant. I think it would be very interesting to have some

discussion on this subject. Mr. Langton, did you tell me you had prepared some remarks?

JOHN LANGTON, New York, N. Y.—This is an exceedingly useful paper. The plant it describes constitutes a long step in advance, and a bold step; and it is pleasant to know that the courage needed to undertake it has been rewarded by success. We have known that Diesel engines have been gradually and carefully developed by at least three manufacturers in Europe to units of large size, and of demonstrated reliability with the fuel oils there used. We knew that in getting the fuel oil into the cylinders and igniting it, European practice had dealt with fuels more difficult than viscous California oil. But where the same viscosity is accompanied by a different composition, the parallel ends at combustion. All contracts for California fuel oil as used for boilers and furnaces in Arizona contain a clause allowing a small quantity of sand. This oil is the only form of fuel we can depend upon, both in quantity and price, on which to base reliable estimates of the fuel-oil costs of operating Diesel engines in Arizona. As compared with the known conditions of European practice, there remained the serious question whether the sand allowance, together with any other ash content of asphaltum-base California oil, might not form enough grit to cause prohibitive wear in the cylinders. The Tyrone plant has done us the very great service of definitely determining this cardinal matter.

The Tyrone plant has also settled another minor but weighty point that has heretofore been a troublesome item in estimating operating costs; that is, the quantity and cost of lubricating oils consumed. The table of costs in Mr. Legrand's paper shows that lubricating-oil cost is more than 8 per cent. of the fuel-oil cost at Tyrone; and since the average cost of fuel oil at Arizona points is only $1.45, instead of the $1.85 to $1.96 per barrel stated for Tyrone, the figures he gives for lubrication would add not less than 10 per cent. to the fuel-oil cost in an Arizona plant. Considerable as this is, it is only half of what we have heretofore felt obliged to allow in estimating the cost of lubrication. This notable reduction is in part quantity and in part price per gallon. By inadvertence the quantity of lubricating oil and its cost per gallon were omitted from Mr. Legrand's paper. He has given me these figures with permission to state them. The monthly consumption of one engine running continuously is:

175 gal. of cylinder oil at 42c. per gallon.
170 gal. of engine oil at 28c. per gallon.

At the 1,250-b.hp. sea-level rating of these engines, these figures correspond to:

$1.16 cost of lubrication per brake horsepower-year of rating and
1.36 grams of lubricating oil per brake horsepower-hour of rating.

I have stated the quantity of oil in grams in order to compare with the

gures for estimation given by two European manufacturers. They ere:

2 to 3 grams given me by one manufacturer and
2 to 2.5 grams given to Mr. Legrand by another manufacturer in another country.

The experience at Tyrone does not cover the question of what sulphur ontent is permissible in the fuel. I understand from Mr. Legrand that here was less than 1 per cent. sulphur in the fuel oil at Tyrone. In urope, I found that sulphur was a great bugbear, but from such inquiries s I was able to make, and from what others tell me of their inquiries, here is a considerable difference of opinion as to what sulphur content is ermissible. One maker stated explicitly that up to 1 per cent. sulphur ives no trouble in the cylinders, provided there is a suitable quality of ylinder oil, but that 2 per cent. is the practical limit of sulphur with any uality of cylinder oil. On the other hand, another maker referred to heir engines running satisfactorily on Rumania oil with 3 per cent. ulphur. I was informed that the bad effect of sulphur seems to be wear f the cylinder, not by corrosion, but indirectly by destroying the lubrica- ion. It is possible that the low cost of cylinder oil at Tyrone, both in uality and quantity, may be due to the low sulphur. That, however, oes not affect the value and certainty of the Tyrone figures to us in rizona, where the fuel oil is identical with the Tyrone oil.

Thanks to this valuable paper, we can now estimate with confidence oth fuel oil and lubrication costs. Operating labor can be foretold ith a close degree of accuracy, and the minor item of supplies with a ufficient degree of approximation. The interest and amortization harges are definitely fixed, as a matter of opinion or policy based on the onditions of the industry that the power plant serves. There only re- nains the very important item of maintenance. That is not yet de- ermined at Tyrone. But when it is determined for that plant, I fear hat it will be of but slight help to others. In plants consisting of a ew large and costly units, individual good or bad fortune must always be a very large factor in repair costs. In estimating maintenance, all we can do is to try and be on the safe side and allow for a larger cost than we can imagine to be possible. Whether the results show surplus or deficit in this item is a question of good or bad fortune at the plant, and of whether the estimator is a pessimist or an optimist.

S. J. Kidder, Mogollon, N. M.—There is one question I would like to ask as to the cost of lubrication per brake horsepower year. It was stated that it was based on 1,250 hp. as the load of the engine.

John Langton.—The rating of the engine. I might have said that comparative quantities of lubricating oil per horsepower are best estimated on the rating of the engines running, rather than on the load they may be carrying. The variation in total cost of lubrication at different loads is probably comparatively small.

Ore-Drawing Tests and the Resulting Mining Method of Inspiration Consolidated Copper Co.

BY GEORGE R. LEHMAN, B. A., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

THE Inspiration Consolidated Copper Co. had an orebody at Miami, Ariz., of close to 100,000,000 tons of low-grade copper ore, and the method of mining this ore most profitably was of great importance. The selection of a method was practically limited to a "top-slicing," a "shrinkage-stope" or a "caving" method. The "top-slicing" method would give a high extraction of the developed ore, at a high mining cost, and the "shrinkage-stope" or "caving" methods a lower extraction of the developed ore, at a lower mining cost.

ORE-DRAWING TESTS

To decide on the method to be used it was necessary to know about what extraction of the ore could be obtained by the "shrinkage-stope" or "caving" methods. As no definite information regarding the question was available at the time, C. E. Mills, the general manager, thought that some information could be obtained by experimenting in drawing ore, covered with capping, from an experimental box. Some work had already been done along these lines by W. C. Browning for the Inspiration Copper Co. and by myself for the Live Oak Development Co.

The idea was to represent as nearly as possible, in the experimental box, the conditions within an area of caved stopes of caved ore all ready for drawing. That is, the area was supposed to represent a semibroken mass with the capping above it in the same condition and ready to follow the ore downward as it was drawn out of the chutes under the ore. To represent the above condition, in the experimental box, crushed ore from the mine was placed in it to a given height, and red barren capping from the mine was placed on top of the ore.

The Experimental Box

The experimental box shown in Fig. 1 was made of wood and glass. The glass was used for the sides so that the drawing action of the outside row of chutes could be seen and noted, the latter being cut in the center by the glass sides. The length of the box was 30 in., width 20 in. and the

height 25 in., and built to a scale of 1 in. = 5 ft., the box represented an area of 150 by 100 ft. and a height of 125 ft. within the mine. The box, as shown in Fig. 1, was divided in the center and only half was used for each test. The bottom was bored with ½-in. holes, representing chutes, 1¼ in. center to center, thus indicating chutes of 2½ ft. diameter spaced on 6¼-ft. centers according to the scale. Thus the ore could be represented as drawn at either 6¼, 8⅞, or 12½-ft. chute intervals. The holes were fitted with wooden plugs through which wire nails were driven, so

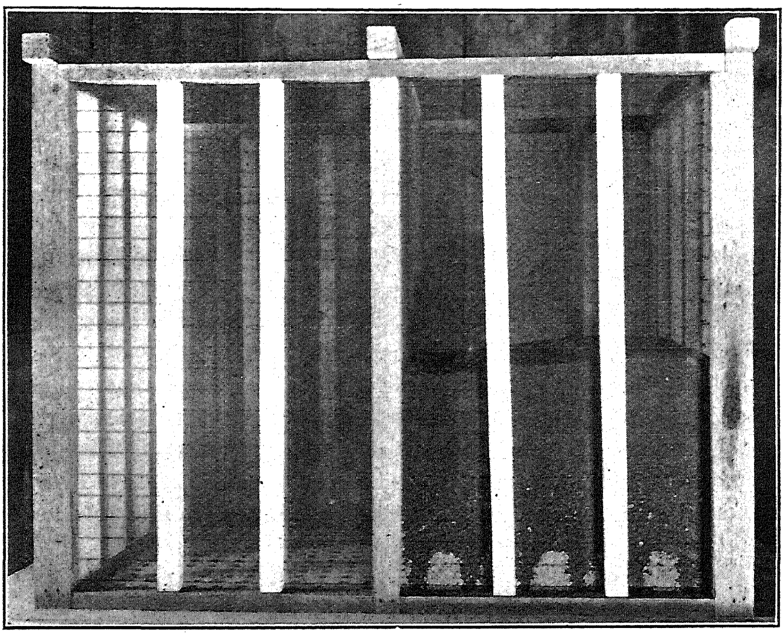

Fig. 1.—Experimental Box Filled with Crushed Ore and Capping; Drawn through Holes in the Bottom Representing Ore Chutes.
Scale, 1 In. of Box=5 Ft. of Mine.

that when the plugs were in place the heads of the nails projected just the least bit above the inner bottom of the box. The nail was used to clear the chute when it "hung up."

Conditions of Tests

In all, 16 tests or experiments were made in a period of 6 months beginning in the latter part of the year 1913. In all the tests the rock representing capping was nearly barren, containing only from 0.03 per cent. to 0.08 per cent. copper. The color of the capping was a dark red and that of the ore a light gray, thus giving a sharp and definite line between them, as clearly seen in Fig. 1.

In the first two experiments, weights and assays were not used. The

ore in the box and products drawn from the chutes were measured by volume. They were run to get an idea, in a short time, of what could be done. In all the other tests the ore was weighed, sampled and assayed for copper, as were the chute products drawn during the test.

Tests Nos. 3 and 4 were made with fine and coarse ore mixed, the capping being a similar mixture of coarse and fine material. The conditions of test No. 3 represented 35 ft. of ore and 35 ft. of "capping" and those of test No. 4 represented 70 ft. of ore and 55 ft. of capping. Both tests were drawn at the closest chute interval, 6¼ ft. In both tests ore was drawn from one chute equivalent to about 10 tons, on the scale adopted, and then 10 tons from the next chute, and so on until 10 tons had been drawn from each chute. The drawing of all the chutes was then repeated, 10 tons at a time, until capping appeared in each chute. The total chute-drawings product was then weighed, sampled, assayed and called the clean-ore product. Drawing was then continued as before until each chute showed that it was running one-quarter capping. To determine this point, a sample mixture was kept in a test-tube and the chute product compared with it. This product was then weighed, sampled and assayed. Again the drawing continued until each chute showed about one-half capping. This point was also determined by comparing with a sample mixture and this product also weighed, sampled and assayed. The tests were concluded by drawing 10 tons from each chute in rotation, weighing, sampling and assaying the product. The assay value for the combined products of the tests was calculated, as the portions removed for assay could not be returned. Details of test No. 3 are given in Table 1.

TABLE 1.—*Details of Test No.* 3

Weights and assays—35 ft. of ore and 35 ft. of capping.
Ore and capping—coarse and fine mixed.
Largest boulder—1 ft. diameter.
Smallest particle—powder.
Drawn at 6¼-ft. centers—10 tons at one drawing.

Ore in box 127.50 lb............ 1.96 per cent. copper
Capping .... 0.08 per cent. copper

Products	Pounds	Per Cent.	Per Cent. Cu
(1) Drawn clean to capping.	81.00	63.52	1.96
(2) Drawn clean to ¼ capping	8.50	6.67	1.36
(3) Drawn clean to ½ capping	8 50	6.67	1.10
(4) Drawn 10 tons each chute.	11.00	8.62	0.79
Totals and average	109.00	85.48	1.74

TABLE 1.—*Details of Test No. 3 (Continued)*

Products	Per Cent Total	Grade, Per Cent. Copper	Product		Per Cent. Total Cu Recovered
			Per Cent. Ore	Per Cent. Capping	
Clean ore.... . (1)	63.52	1.96	63.52	.....	63.52
Ore and capping (2)	6.67	1 36	4.54	2.13	4.54
Ore. ....(3)	6.67	1.10	3.62	3.05	3.62
	76.86	1.83	71.68	5.18	71.68

Estimate Applied to above Figures to Find Final Product of 1 Per Cent. Copper and Applied to Ores Below

Grade, Per Cent. Copper	Extraction, Per Cent. of Total	Grade of Product, Per Cent. Copper	Per Cent of Total		Per Cent of Product	
			Ore	Capping	Ore	Capping
2.00	79.500	1.842	72.96	6.54	91.77	8.23
1.90	78.340	1 763	72.44	5.90	92.47	7.53
1.80	77.180	1 681	71.84	5.34	93.08	6.92
1.70	75.490	1.605	71.07	4.42	94.14	5.86
1.60	73.490	1.529	70.06	3.43	95.33	4.67
1.50	71.490	1.449	68.92	2.57	96.41	3.59
1.40	69.290	1.367	67.54	1.75	97.48	2.52
1.30	67.290	1.283	66.35	0.94	98.61	1.39
1.20	65.290	1.195	65.02	0.27	99 58	0.42

Above Applied to Mining—25 Per Cent by Development

2 00	84.625	1.889	79.81	4.81	94.31	5.69
1.90	83.755	1.804	79.41	4.35	94.81	5.19
1.80	82.885	1.717	78.95	3.93	95.26	4.74
1.70	81.617	1.634	78.35	3.26	96.00	4.00
1.60	80.117	1.551	77.59	2.53	96.84	3.16
1.50	78.617	1.465	76.72	1.89	97.59	2.41
1.40	77.117	1.378	75.86	1.26	98.37	1.63
1.30	75.467	1.289	74.80	0.66	99.12	0.88
1.20	73.967	1.197	73.78	0.19	99.74	0.26

Besides estimating the extractions for different grades of ore, based on the experiments, the extractions for different grades of ore applied to mining were calculated, as shown in the lower part of Table 1, assuming 25 per cent. of the ore extracted by development and other work before the ore caves, when mining 35 ft. of ore, and 15 per cent. when mining 70 ft. of ore. For each grade, 1 per cent. copper was taken as the limit of commercial product. This point had to be interpolated from the product just above 1 per cent. and the product just below 1 per cent.

TABLE 2.—*Comparison of Extractions Based on 1.80 Per Cent. Copper Ore*

Test No.	Condition of Ore and Capping	Method of Drawing	Distance between Chute Centers, Feet	Depth of Ore, Feet	Extraction, Per Cent Total	Grade of Product, Per Cent Copper	Per Cent. Total		Per Cent Product	
							Ore	Capping	Ore	Capping
3	Coarse and fine mixed.	Clean to cap—¼ cap. ½ cap.—10 tons each chute.	6¼	35	77.180	1.681	71.81	5.31	93.08	6.92
4	Coarse and fine mixed.	Clean to cap—¼ cap ½ cap.—10 tons each chute	6¼	70	93.950	1.705	88.76	5.19	94.48	5.52
5	Coarse, 4 to 12 in	Clean to cap—¼ cap. ½ cap.—10 tons each chute	6¼	35	86.400	1.717	82.30	4.10	95.26	4.74
6	Coarse, 4 to 12 in	Clean to cap—¼ cap. ½ cap.—10 tons each chute.	6¼	70	95.030	1.765	93.13	1.90	98.00	2.00
7	Coarse, 4 to 12 in.	Clean to cap—¼ cap ½ cap.—10 tons each chute.	8⅞	35	78.570	1.717	74.85	3.72	95.26	4.74
8	Coarse, 4 to 12 in	Clean to cap—¼ cap ½ cap.—10 tons each chute	8⅞	70	89.550	1.764	87.71	1.84	97.94	2.06
10	Coarse, 4 to 12 in.	Clean to cap.—¼ cap. ½ cap.—10 tons each chute.	12½	70	85.110	1.741	82.24	2.87	96.63	3.37
11	Coarse, 4 to 12 in.	Clean to cap—¼ cap. ½ cap.—10 tons each chute	12½	35	67.390	1.707	63.81	3.58	94.69	5.31
12	Coarse, 4 to 12 in	Direct to cap.—¼ cap—½ cap. each chute.	8⅞	35	84.290	1.708	79.86	4.43	94.74	5.26
13	Coarse, 4 to 12 in.	Direct to cap.—followed by 10 tons each chute.	8⅞	35	83.980	1.703	79.33	4.65	94.46	5.54
14	Ore coarse, 12 to 30 in. Capping fine, 2 in. to powder.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	76.050	1.600	67.36	8.69	88.57	11.43
15	Ore fine, 2 in. to powder. Capping coarse, 12 to 18 in	25 tons each chute to capping—then 15 tons each chute	8⅞	35	80.040	1.564	69.24	10.80	86.51	13.49
16	Ore and capping, fine.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	53.940	1.635	48.85	5.09	90.57	9.43

TABLE 2.—(*Continued*)

Above Tests Applied to Mining 1.80 Per Cent Ore—25 Per Cent by Development for 35 Ft; 15 Per Cent for 70 Ft

3	Coarse and fine mixed.	Clean to cap—¼ cap ½ cap—10 tons each chute	6¼	35	82.885	1.717	78.95	3.93	95.26	4.71
4	Coarse and fine mixed	Clean to cap—¼ cap. ½ cap—10 tons each chute	6¼	70	91.857	1.720	90.52	4.33	95.13	4.57
5	Coarse, 4 to 12 in.	Clean to cap.—¼ cap ½ cap—10 tons each chute.	6¼	35	89.800	1.740	86.72	3.08	96.57	3.43
6	Coarse, 4 to 12 in.	Clean to cap—¼ cap. ½ cap—10 tons each chute	6¼	70	95.775	1.770	94.13	1.65	98.28	1.72
7	Coarse, 4 to 12 in.	Clean to cap—¼ cap. ½ cap—10 tons each chute	8⅞	35	83.927	1.742	81.15	2.78	96.69	3.31
8	Coarse, 4 to 12 in.	Clean to cap—¼ cap. ½ cap—10 tons each chute	8⅞	70	91.117	1.770	89.55	1.57	98.28	1.72
10	Coarse, 4 to 12 in.	Clean to cap—¼ cap. ½ cap—10 tons each chute.	12½	70	87.343	1.751	84.90	2.44	97.20	2.80
11	Coarse, 4 to 12 in	Clean to cap—¼ cap. ½ cap—10 tons each chute	12½	35	75.542	1.738	72.87	2.67	96.46	3.54
12	Coarse, 4 to 12 in	Direct to cap—¼ cap—½ cap—each chute.	8⅞	35	88.217	1.734	84.89	3.33	96.23	3.77
13	Coarse, 4 to 12 in.	Direct to cap followed by 10 tons each chute	8⅞	35	87.985	1.731	84.52	3.47	96.06	3.94
14	Coarse ore, fine capping.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	82.037	1.661	75.53	6.51	92.06	7.94
15	Fine ore, coarse capping	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	85.030	1.633	76.92	8.11	90.46	9.54
16	Ore and capping, fine.	25 tons each chute to capping—then 15 tons each chute.	8⅞	35	65.455	1.698	61.64	3.82	94.17	5.83
		Average			85.38	1.723	81.63	3.75	95.16	4.39

This way of estimating the results possible in mining a 1.80 per cent. copper ore, according to test No. 3 indicates a total extraction of 82.88 per cent. of 1.72 per cent. ore, 78.95 per cent. of which was original ore, the chute product being 95.26 per cent. ore and 4.74 per cent. capping.

In the next two tests, Nos. 5 and 6, the fines were screened out of the ore and capping, making both ore and capping represent, on the scale of 1 in. = 5 ft., coarse chunks of 4 to 12 in. diameter. The chute interval and method of drawing were the same as in tests Nos. 3 and 4. Test No. 5 represented drawing 35 ft. of ore and test No. 6 represented 70 ft. of ore. Compared with tests Nos. 3 and 4, as shown in Table 2, both tests show higher extractions.

Tests Nos. 7 and 8 were duplicates of 5 and 6 except that the chute interval was 8⅞ ft. instead of 6¼ ft. These two tests show lower extractions than Nos. 5 and 6.

Tests Nos. 11 and 10 were the same as Nos. 5, 6, 7 and 8 except that the chute interval was 12½ ft. As will be noted in Table 2, still lower extractions were obtained with these two tests, showing conclusively that the greater the distance between chutes the less the ore extraction.

To determine the effect on extraction if no care is taken as to the amount drawn at one time, two tests, Nos. 12 and 13, were run. They were both made with conditions representing 35 ft. of ore and 35 ft. of capping, drawn at a chute interval of 8⅞ ft. In these two tests each chute was drawn direct to capping in rotation, then direct to one-fourth capping and then direct to one-half capping. Comparing these two tests with No. 7, with which all conditions were identical, except the method of drawing, it was found that in No. 7 a greater extraction of clean ore, before capping appeared, was obtained, but that in Nos. 12 and 13 a greater total extraction was obtained to the limit of 1 per cent. copper, as shown in Table 2.

Tests Nos. 14, 15 and 16 were special experiments made with conditions representing a chute interval of 8⅞ ft. and the column of ore 35 ft. In No. 14 the ore was coarse, 12 to 30 in. diameter, and the capping fine; a low extraction was obtained. Test No. 15 was the reverse of No. 14, the ore was fine and the capping coarse; the extraction was but very little higher than in No. 14. The last test, No. 16, was made with conditions representing both ore and capping fine. This apparently was the worst condition, the extraction being very low.

Conclusions Derived from Experiments

The conclusions derived from the experiments were as follows: (1) The less the distance between chute centers, the greater the extraction; (2) the higher the ore column, the greater the extraction; (3) the less the amount drawn at a time from each chute, uniformly, the greater the extraction of clean ore before the capping appeared, but not the greater the total

extraction to a 1 per cent. copper final product; (4) the coarser the ore and capping without fines, the greater the extraction; (5) the higher the grade of ore, the greater the extraction, when a uniforn limit of grade is allowed at the end of the chute drawing. (6) Applied to mining, the results of the experiments indicate that the greater the amount extracted by development and undercutting, the greater will be the total extraction. This is on the assumption that when the ore caves it occupies the same volume as in place.

Some suggestions deduced from these experiments considered desirable to carry out in mining were: (1) Chute centers for drawing caved ore should be as near together as the ground will allow; (2) ore should be caved in as high columns as practicable; (3) when capping appears at the chutes, no more should be drawn unless the bottom slice is being mined.

While all of the tests described were made with a barren capping, in the actual mining of a sulphide orebody having mixed or oxidized ore over it in place of barren capping, the recovery of sulphide ore would be considerably increased over the experimental results, provided the mixed or oxidized ore could be treated at a profit.

In deciding what extraction could be expected by caving from 35 ft. to 70 ft. of ore in average caving ground, the results obtained in tests Nos. 10 and 11, as detailed in Table 2, were considered conservative, provided the chutes were closely watched and ore of below the commercial limit not drawn. If the ground under the caved ore, the location for raises and chutes, is hard and strong, results of tests Nos. 7 and 8 could be taken as obtainable. This opinion is based on the supposition that the capping will break up in about the same way as the ore and not into a fine powdery sand.

Method of Mining

The Inspiration company's method of mining is one of the so-called Caving systems whereby the ore is caused to cave and crush itself, thus reducing to a minimum the blasting and handling. It is a modification of the method introduced by Felix McDonald at the Ohio Copper Co.'s mines in Utah, and was put into operation at the Inspiration under his supervision. The method consists, essentially, of undercutting the ore (taking out a horizontal slice), allowing the ore above to cave and crush, and drawing off the crushed ore through small inclined raises driven under the caved ore, into main inclined raises that lead down to the haulage-drift chutes.

Preliminary Development Work

By referring to the accompanying illustrations showing the progressive steps of mine development, a good understanding of the method will be obtained.

After shafts have been sunk, the haulage drifts are driven under the ore of the section or sections to be mined at intervals of 100 ft. as shown in Fig. 2. These drifts are of large size, being 9 ft. wide at the base of the rails, 7½ ft. wide at the cap, and 7½ ft. high above the rail base. Where timbered, the above refer to inside timber dimensions.

After the haulage drifts are in, or during their driving, "pony" sets are put up every 25 ft. along the drifts. The "pony" set is about 5 ft. high, placed on top of the regular drift set and is the place from which the car loaders operate the chute gates discharging into the haulage cars.

Inclined raises are then started from the "pony" sets at an inclination of from 50° to 54°, depending on whether the sublevels are to be

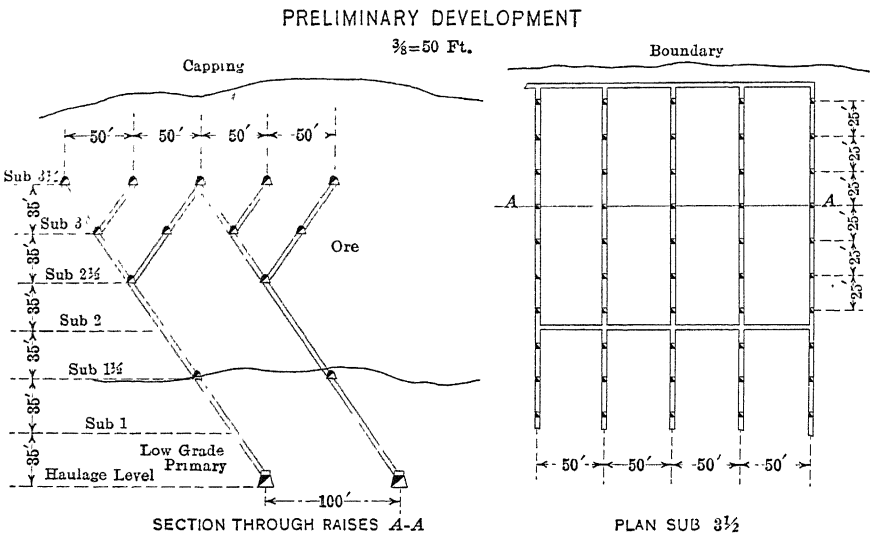

FIG. 2.—PRELIMINARY DEVELOPMENT UNDERTAKEN BY DRIVING INCLINED RAISES FROM HAULAGE-LEVEL DRIFTS.

30 or 35 ft. vertically apart, in a plane at right angles to the haulage drifts. When these raises are up from 10 to 15 ft., chutes are built at the "pony" sets and ore gates of steel are installed.

The raises are then advanced until the first sublevel is reached. "Sub" drifts are then started, from the first raises to reach the required level, and are driven parallel to the haulage drifts. These drifts break into the other raises every 25 ft., as they advance, thus cutting down the cost of handling the drift "muck." The raises are then again advanced to the next sublevel where drifting is to start. The drifts on this level are then driven, meeting the raises as they advance. All of these development raises are about 4 ft. in diameter, and the subdrifts 6 by 7 ft.

This method of raising and sublevel drifting is continued until the

height is reached where the first undercut is to be made, and the drifts are driven as before.

The first undercut sublevel will be located below the top of the ore, at about the height at which it is intended to cave. When the preliminary development is finished, the sublevel drifts on the first undercutting level are 50 ft. apart, as they are on the "sub" just below, as shown in Fig. 2. On all the "subs" below, the drifts are 100 ft. apart. All the drifts on each "sub" are also connected with inclined raises every 25 ft. Cross drifts, connecting all the drifts on each "sub," are also driven at intervals of about 150 ft.

The next work, before undercutting is commenced, is to drive on the level to be undercut other "sub" drifts, between and parallel to those

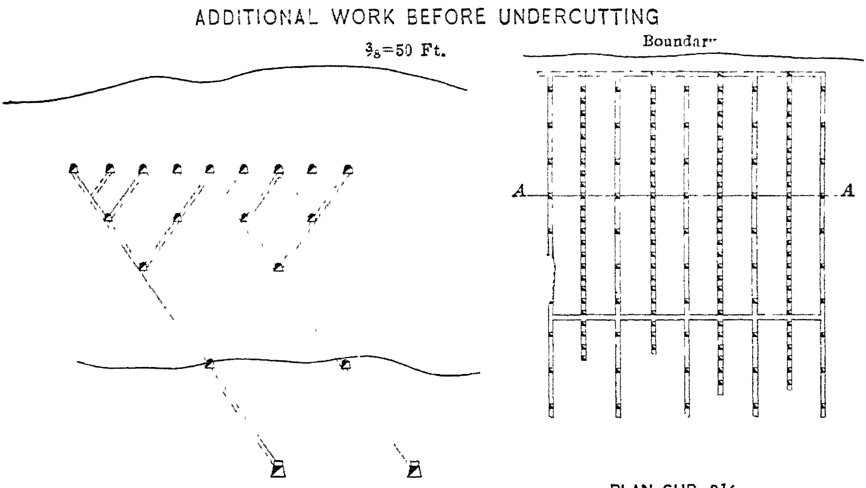

Fig. 3.—Preceding Undercutting, the Topmost Level is Developed by Drifts Spaced on 25-Ft. Centers and Connected with the Branch, or "Finger Raises."

already driven, as best shown by Fig. 3, making the drifts on this "sub" 25 ft. center to center. These drifts, as they advance, meet branch raises of the main inclined raises at 12½- or 25-ft. intervals as desired. These small branch raises, shown in Fig. 4, are called "finger raises," many of which are put up just before or ahead of the undercutting operations.

Undercutting

Undercutting the ore is accomplished by starting at a cross drift, on the boundary of the section to be mined, and in retreating from that cross drift, as best shown in the plan of Fig. 5, drilling deep holes, at nearly right angles to the drifts, into the pillars between them, and blasting out the ground. Three holes are drilled in each side at different angles in

the same vertical plane, with one hole in the back of the drifts, thus making seven holes to the round. For this work a water-hammer one-man drill with large steel is used. The holes are from 8 to 10 ft. in depth. Usually the rounds are blasted one at a time, the undercutting receding from the caved ground.

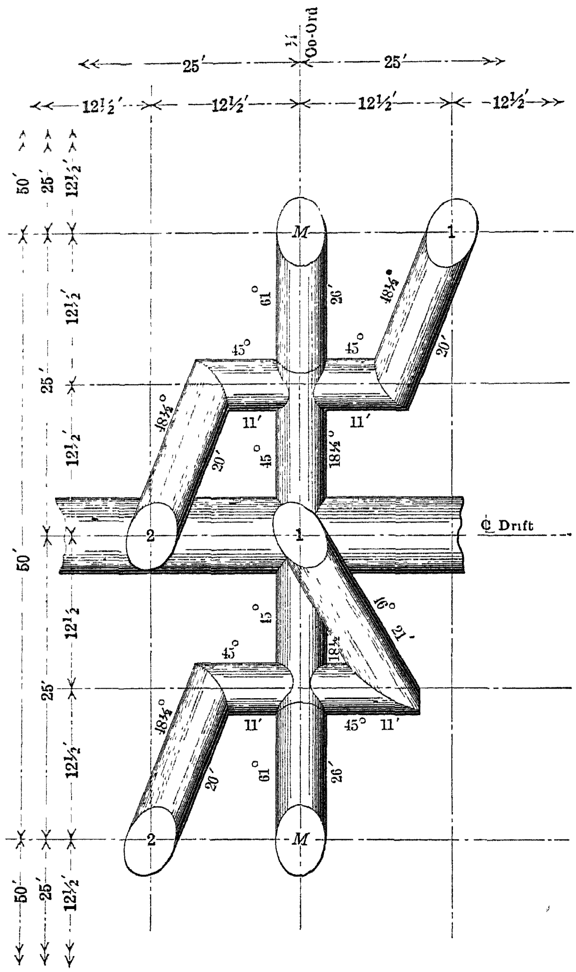

FIG. 4.—"FINGER RAISES", AN IDEAL TOP VIEW SHOWING THEIR CONNECTIONS WITH MAIN RAISES, *M*, AND SUBLEVEL DRIFT.

Ore from the undercut is drawn through the "finger raises," in each of which has been built an ordinary board chute to control the drawing. The "finger-raise" chutes are located about 4 or 5 ft. below the sublevel, so that they will not be blasted in shooting the undercutting holes.

While in some ground, the ore begins to cave as soon as it is undercut, in hard ground caving does not start until the undercutting has receded a considerable distance.

Ore Drawing

After the ore caves it is drawn off as desired. Large boulders that will not pass the raise chutes are blasted. In drawing off the ore, the chute "tappers" work in pairs. One "tapper" goes up a raise, from the "grizzly sub," which is the first "sub" below the undercutting level, opens a chute gate and draws the ore, while the other "tapper" works the ore through the grizzly on the "sub" below.

All grizzlies have about 1-ft. openings and are made of timber or steel rails. They are placed over all raises on the "grizzly sub" and are set at right angles to the drift. Pieces of ore too large to pass the grizzly are broken with an 8-lb. hammer by the "tapper" tending the grizzly.

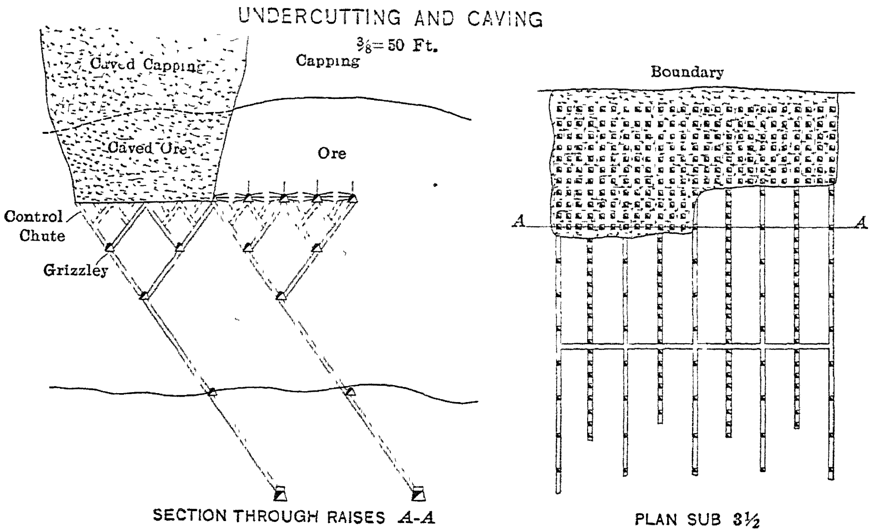

FIG. 5.—STARTING UNDERCUTTING BY BLASTING DOWN THE PILLARS BETWEEN SUBLEVEL DRIFTS.

After passing the grizzlies, the ore falls into the main raises and down to the haulage-drift chutes where it is loaded into 5-ton cars and hauled in trains, of 15 to 20 cars, to the shaft bins.

The second undercut can be located at one, or any number of sub-levels below the first undercut, according to the height of ore which it is desired to cave. It could be located at the bottom of the ore if desired. In fact, the first undercut could be located at the bottom of the ore if it were known that the orebody to be mined could be caved throughout its total height.

Underground Haulage and Hoisting Arrangements

At the Inspiration mine the ore trains are hauled by compressed-air locomotives built by the H. K. Porter Co. of Pittsburgh. Arrived at the

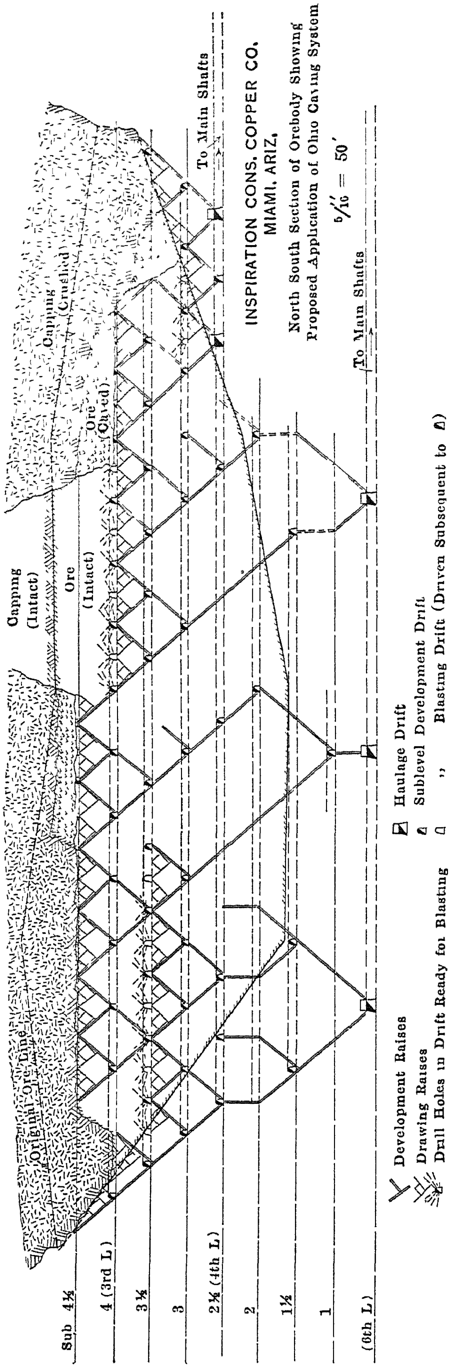

FIG. 6.—ARRANGEMENT OF MINE LEVELS AND RAISES UTILIZED IN INSPIRATION CAVING SYSTEM.

shaft, the cars are tippled into the shaft bins by barrel-shaped tipples, five cars at a time. From the shaft bins the ore is loaded automatically into 12½-ton skips and hoisted by electrically driven automatic hoists to the steel bins on the surface.

For hoisting ore, two main shafts 101 ft. apart are used. Both shafts have three compartments, of which two in each shaft are skipways. The third compartment in one shaft contains the ladderway, pipes and electric conduits, while in the other shaft the third compartment contains an Otis double-deck elevator for hoisting and lowering men. Both shafts, the underground bins, and the stations are all lined with reinforced concrete.

Only two levels are used for hauling ore, the 4th and 6th. The vertical distance between these two levels is 130 ft. A good idea of the arrangement of the mine is obtained from Fig. 6. On the 6th level the shafts are connected by two drifts, one for each shaft. In each drift there is a tipple over the bin at the shaft. The arrangement of the drifts, bins, tipples and stations is symmetrical with a center line between the two shafts so that the tipples are operated by one man from a central location.

On the 4th level one double-track drift passes between the two shafts and is connected to the shaft stations by a small cross drift. The ore is dumped by one tipple, on this level, into a small bin which is connected with the 6th level bins by a concrete-lined inclined raise. This arrangement makes necessary only one loading level for the skips.

The Inspiration Consolidated Copper Co. commenced underground development on a large scale toward the latter part of 1913 but did not start regular mining operations until August, 1915. The tonnage mined gradually increased as the concentrator was able to handle it until at present (June, 1916) an average tonnage of 16,700 tons is being mined daily.

The cost of mining is now 60 c. per ton, including 20 c. for development and all fixed charges.

The percentage of ore and copper extraction obtained by the Inspiration method cannot be determined positively until some section has been entirely mined from top to bottom of the ore. But based on results in six sections of the mine from the first undercut to the capping estimated to contain 1,886,450 tons, and which have been almost completely drawn to capping, the ore extraction is 102.44 per cent. and the recovery of copper 86.52 per cent. The second undercut is expected to increase the copper recovery.

Shaft Sinking through Soft Material

BY EDWARD A. SAYRE,* E. M., DES MOINES, IOWA

(Arizona Meeting, September, 1916)

In shaft sinking for coal mines, the cost item greatly influences the method adopted. This holds true especially when soft material must be traversed. The average life of a coal mine is short. This is due either to the limited area of the coal basin or to the great expense of maintenance and haulage underground when the entries are extended a considerable distance from the hoisting shaft. Therefore, the engineer in opening a coal mine is confronted with the fact that the cost must be kept within certain bounds else the venture will prove unprofitable. For this reason he may be prohibited from adopting certain safe, but expensive, methods that are used in sinking caissons for foundations, driving transportation tunnels, or other work of permanent construction.

A common method of sinking through difficult ground employs the aid of a steel shoe pushed ahead of the shaft timbers. Another is the drop-shaft method. These two methods were used in sinking the main shaft and the air shaft of the Eagle No. 3 mine, at Des Moines, Ia., and the following data show the relative success and cost of the two methods of sinking under the same conditions.

A drill log of the material to be penetrated was as follows (see Plate 1, Fig. 1):

	Thickness, Ft.	In.	Total Depth, Ft.	In.
Drift	38		38	
Sand	2		40	
Drift	2	6	42	6
Jointed clay	5	6	48	
Drift, firm	10	6	58	6
Clay, with sand streaks	3		61	6
Sand	12		73	6
Shale, with two thin beds of coal and thin strata of rock	86	6	160	
Rock	5		165	
Coal	3	9	168	9

Main Shaft Sunk by Steel-Shoe Method

This record shows the first 73 ft. to be of drift material, the lower 12 or 15 ft. of which was sand. The remaining 92 ft. to the coal was shale and rock, and offered no trouble in sinking.

* General Manager, Eagle Coal & Mining Co.

The general equipment, purchased from a company that had twice attempted to sink on this property and failed, consisted of one 80-hp. boiler, a hoisting engine, one No. 5 horizontal and one No. 6 vertical

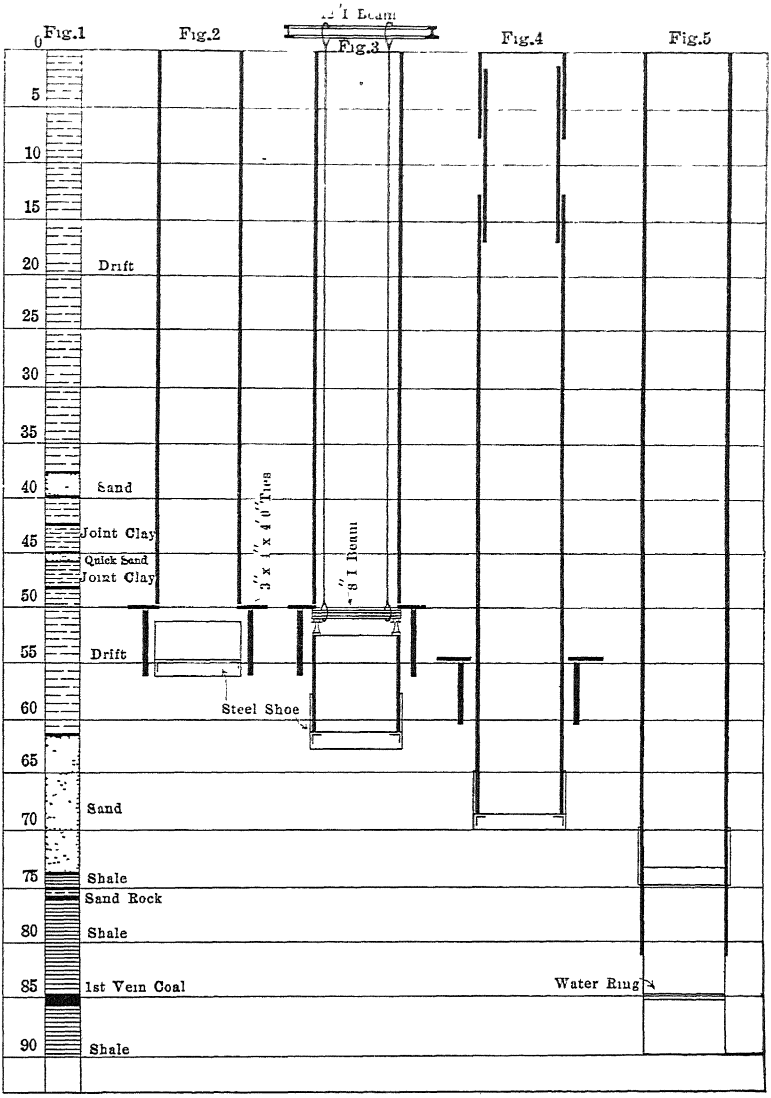

Plate 1.

Cameron pump, a dump car, hoisting buckets, etc. The men employed were a day and a night engineer, one timberman, and three sinking crews, each consisting of three sinkers and one top man. Work was

continuous 24 hr. a day until the sand was reached, when it was cut to two shifts, the remaining time being used in repacking pumps, etc.

The excavation was made 8 by 14 ft. The curbing for the first 50 ft. consisted of 2 by 4 and 2 by 6 timber laid flatwise, alternately with a quarter shaft bratticed off with 3 by 12 material, and the hoisting compartments divided by 4 by 6 buntons.

No trouble was experienced in sinking until the bed of sand at the 38-ft. level was encountered. This sand carried a small amount of water and, as a shaft protection, a concrete ring was put in behind the curb. The next trouble was experienced from the 42- to 48-ft. level. It had been the intention to assemble the shoe at this level, but what appeared to be a firm green clay proved to be jointed clay with a seam of quicksand through it and the exposed wall would not stand. Temporary 2 by 12 curbing was put in place and then regular curbing carried on down.

At the 50-ft. level, in order to safeguard the assembling of the steel shoe, 125 wooden mine ties, 3 by 4 in. by 4 ft. long, were sledged into the clay just below the curb as shown in Plate 2. Steel angles, 6 by 4 in., were fastened with lag screws to the under side of the ties. An excavation 6 ft. deep was then made, undercutting the ties 16 in., and a false curb of 2 by 6 boards built. The shoe was then assembled at this point.

The shoe consisted of four sheets of 3/8-in. steel, two 8 ft. long by 5 ft. high and two 14 ft. long by 5 ft. high, with 3-in. angles at the corners, and a 6-in. pressure angle placed 18 in. from the bottom of the shoe. The curbing was then built inside the shoe to a height permitting the jack screws to be placed against the curb timbers of the shaft, as shown in Plate 2.

The clay was then excavated, the shoe lowered into the sand, and pumping started. The sand carried about 60 gal. of water per minute, which would rise about 18 ft. above the sand when pumping was stopped.

In order to avoid the formation of cavities behind the curb, as little sand as possible was excavated. To prevent an inrush of sand under the shoe, at least 1½ to 2 ft. of sand had to be left within the shoe. The method of sinking was to agitate the sand at the bottom of the shoe and force the shoe through it by means of the jack screws bearing against the shoe and the curb of the shaft. Two methods were used to agitate the sand. First, while the men could reach the bottom of the shoe, they stirred the sand with spades. By this method the shoe was lowered about 18 in. in the sand. Later the pumping system shown in Plate 2 was used.

In this second process the discharge of the No. 6 pump could, when desired, be sent through five 3/4-in. pipes, as indicated in the drawing, and these five jets of water could be played upon the sand at the bottom of

the shoe. It was found that these jets would agitate the sand sufficiently to permit the jack screws to push the shoe down, except when the sand was at too high a level inside the shoe, in which case sand would have to be excavated before the jet process could be resumed. In using this process the men would stir the sand with the jets for about 10 min., and then tighten the jack screws. When sufficient space was obtained between the timbers in the shoe and the curbing above, the jack

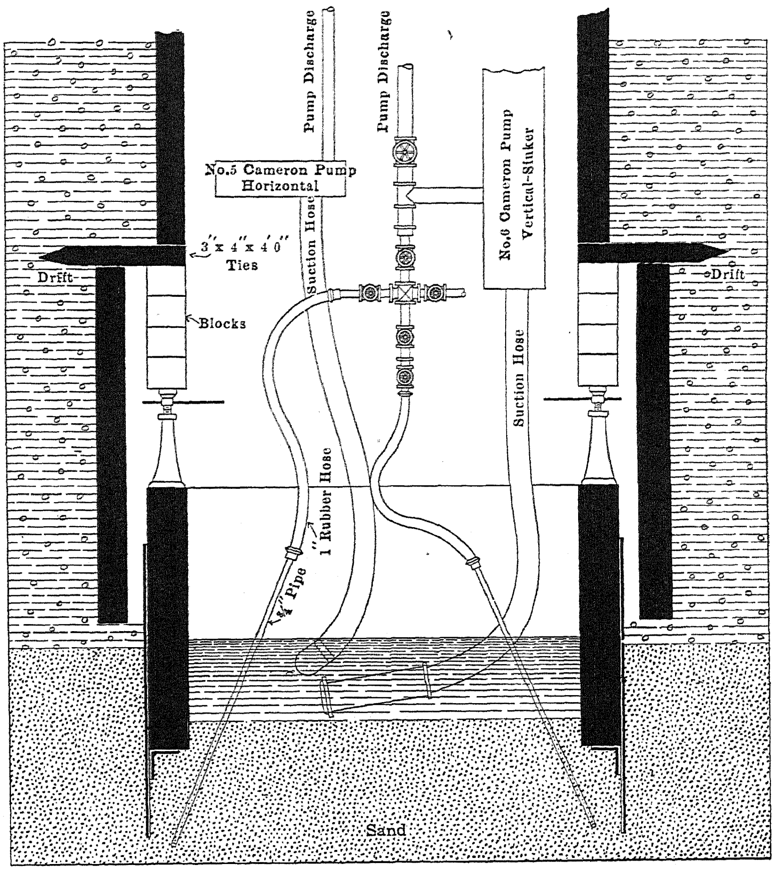

PLATE 2.—DETAILED SKETCH OF SINKING SHOE, JACK SCREWS AND PUMPS.

screws of one side were removed, and sets of timber put in place on top of the timbers in the shoe, and then the other jack screws were moved. An advance of 18 in. per day was exceptional by this method. More often it was less than a foot.

Necessarily some sand had to be excavated at times. This had a tendency to cave the dirt around the shaft, which in turn caused an excessive down pressure and broke the curbing apart a number of times.

An attempt was made to overcome this trouble by supporting the curb with I-beams and cables from the surface as shown in Plate 1, Fig. 3. Ten 12-in. I-beams were supported on cribs at the surface, and ten 8-in. I-beams were swung under the 4 by 6 angle below the ties, these beams being connected by twenty ⅞-in. steel cables. When the sinking was resumed, the curbing continued to break, the I-beams bent, and two cables were broken. Since it appeared impossible to hold the curb, it was decided to timber the shaft solid from the 8-ft. level (where the most uniform break occurred) to the bottom of the shoe, and then drop this portion of the shaft through the remaining 5 ft. of sand.

To do this, the ties were driven back into the wall and solid timbering put in between the shoe and the upper curbing of the shaft. The entire shaft curbing from the 8-ft. level down was then tied together with 2 by 6-in. stringers. At the 8-ft. level, 2 by 4-in. by 16-ft. planks were spiked to the lower curb, the upper ends projecting above the break as shown in Plate 1, Fig. 4, preventing the loose material from falling down the shaft. The jet system, with occasional excavations of sand, was resumed, and the shoe, with 50 ft. of curbing, was lowered through the sand.

In landing the shaft on the solid, seven boulders from 1 to 2 ft. in diameter were encountered. Six of these were under the cutting edge of the shoe, and were removed only after being broken up by means of a long chisel and sledge.

While lowering the shaft through the sand, the upper timbers of the shaft buckled 18 in. out of line. This necessitated retimbering of the shaft from the top of the sand to the surface, an expensive undertaking because the old timbers had to be cut out and replaced in sections.

On completion of the retimbering, sinking through the shale was commenced. Three shifts of four sinkers each were used with an average daily advance of 5 ft. The only problem involved in sinking through the shale was the elimination of the water, a considerable proportion of which was choked off when the solid was reached. This water was taken care of by placing a water ring at the 85-ft. level with a pump located at that point to elevate the water to the surface as shown in Plate 1, Fig. 5.

Air Shaft Sunk by Drop-Shaft Method

The air shaft was located 350 ft. from the main shaft, and its sinking conditions were similar except that the surface at this point was 10 ft. lower than at the main shaft, making the actual distance to be traversed to the solid 63 ft. 6 in.

The equipment was the same as that used at the main shaft. The air shaft followed a drill hole tapped by an entry from the main shaft, so that most of the water was drained through this drill hole and then

pumped to the surface. Two shifts of three sinkers, one top man, and an engineer were employed.

The size of the excavation was the same as before, 8 by 14 ft. The steel shoe was similar to the one in the main shaft, except that it was 10 ft. high instead of 5 ft. The timbering for the first 30 ft. above the shoe consisted of 4 by 6 laid flat, tied together by lag screws ⅝ in. diameter by 10 in. long, spaced 2 ft. apart. The shaft was divided into three equal compartments by 4 by 6 buntons as shown in Plate 3. The middle com-

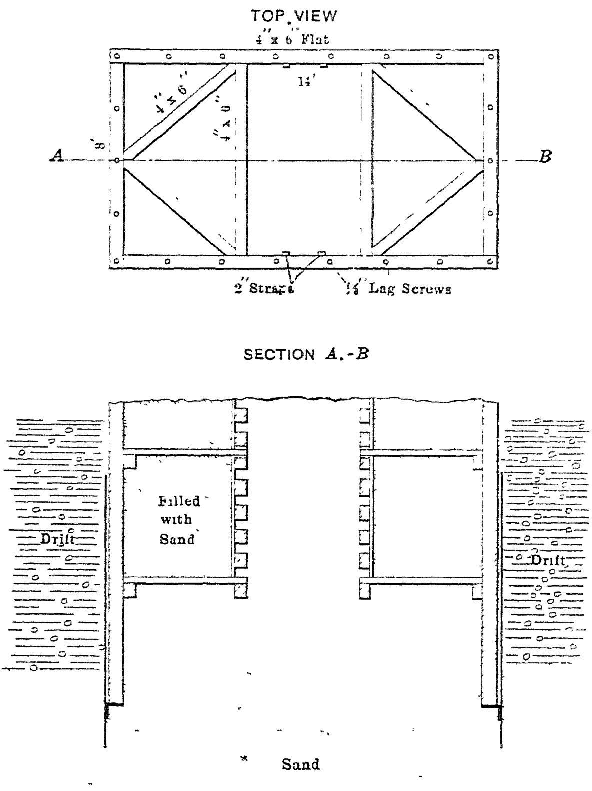

PLATE 3.—METHOD EMPLOYED IN SINKING AIR SHAFT.

partment was left free and was used for hoisting purposes. The end compartments were braced by 4 by 6 extending from the ends of the shaft to the sides. Four steel straps ⅜ in. by 2 in. tied the shaft timbers together from top to bottom.

An excavation 10 ft. deep was first made, the shoe assembled and lined with timbers. The sinking was continued until the shoe was hung up by the friction on the sides. Then a platform was built every 5 ft. in the end compartments. These 5-ft. chambers were filled with sand to give additional weight, and the sinking continued. The ground surrounding

the shoe gradually broke in an oval shape. At one place it was necessary to fire a small charge of powder in order to loosen the ground sufficiently at one end of the shaft.

As the shoe was sunk, timbers were added at the top of the curbing. This method of building the curb at the top is decidedly better than that of adding timbers at the bottom since the timbers are placed much more accurately and expeditiously.

One difficulty experienced in the drop-shaft method was to keep the bottom of the shoe level. When one side of the shoe got lower than the other it kicked the opposite side outward. To right it, the lower side was blocked until the higher side caught up.

The progress through the drift material, until the sand was reached, was slow, much slower than at the main shaft. The drop-shaft went much faster, however, after reaching the sand. In fact, the difficulty at that time was to keep the bottom of the shaft from moving faster than the top. When within 10 in. of the bottom of the sand, the shaft broke apart 20 ft. up from the shoe. This was due to the fact that the movement of the shoe was faster than that of the top of the shaft, and to the insufficient strength of the straps connecting the top and bottom of the curb. At this point (20 ft. above the shoe) the curb separated from 6 to 8 in., and the upper part of the shaft kicked over 9 in. north and east of the lower part. A temporary platform of 8 by 8 timber was put in the end compartments of the shaft at this place, and time given for the upper part of the shaft to settle down before starting the excavation again. The sinking was then continued and the shoe landed on the solid without further difficulty, aside from hitting two small boulders at the bottom of the sand.

As the excavation was larger than necessary for an air shaft, it was decided to cement the shaft for a distance of 28 ft. from the bottom of the shoe, in order to shut off the water. A wall of cement 4 to 8 in. thick was accordingly then constructed.

After the cement was given time to set thoroughly, the excavation was again started in the shale and continued without difficulty to the coal. Sinking through the shale in the air shaft cost slightly more than in the main shaft because work in the mine prevented as careful supervision being given the sinkers in the air shaft.

One difficulty encountered in drop-shaft sinking was in keeping the position of the shaft vertical. At one time this shaft was 2 ft. out of plumb. By regulating the movement at the bottom of the shoe, the shaft partly righted itself, until at the finish, in a total depth of 63 ft. 6 in. to the shale, the bottom of the shaft was 16 in. to the south and 10 in. to the east of the top. Part of this variation was remedied in the cementing of the shaft.

A much larger amount of sand was removed in sinking the air shaft

by the drop-shaft method than in sinking the main shaft. This could be done without danger of a cavity forming, because the surface dirt followed the air shaft down as it descended. When the sinking through the sand was completed, the surface directly surrounding the air shaft had caved to a depth of 15 to 16 ft. and for a distance of 20 ft. in all directions from the shaft. In fact, all the shale that was removed through the remaining 92 ft. to the coal did not fill this space at the surface.

A comparison of the costs of the two shafts is as follows:

	Main Shaft		Air Shaft	
Labor:				
Through drift material . .	$916.80		$789.08	
Through sand.. . .	1,941 80		541.62	
Through shale . ..	1,065.68		1,213.30	
	$3,924.28		$2,544.00	
Superintendence .	600.00		435.50	
Retimbering	1,343.23			
Cementing	. ...		207.68	
Total labor cost..		$5,867.51		$3,187.18
Materials:				
Curbing	$1,878.62		$1,195.17	
Supplies.......	900.14		642.74	
Power, light, water, insurance, etc..............	1,248.50		649 71	
Total curbing, etc., cost.		$4,027.26		$2,487.62
Total costs of shafts		$9,894.77		$5,674.80

Conclusions

In this particular work there was no question about the superiority of the drop-shaft method of sinking. It made a net saving of $4,300 in the total cost of the air shaft compared with the main shaft. A saving of $2,700 was effected in the labor cost, while in the cost of materials, power, etc., the saving was $1,600. A saving in time also resulted, 30 days being required to traverse the sand with the main shaft, while the air shaft was dropped through it in 17 days.

From the results obtained in these two shafts, and from the experience of others in the western interior coal field, we believe that the drop-shaft method of sinking is the safest, most economical, and most successful that can be adopted for sinking through soft material that lies within 100 ft. of the surface. At greater depths a variation of the method can be used by first sinking a larger shaft close to the soft material, and then telescoping a drop-shaft within it.

The Block Method of Top Slicing of the Miami Copper Co.

BY E. G. DEANE,* B. S., E. M., MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

A METHOD of top slicing has been devised at the Miami Copper Co.'s mine at Miami, Ariz., which differs radically in some ways from the customary methods of top slicing.

The area of that section of the orebody in which top slicing is used is about 800 ft. square. The ore, while for the most part soft, is, nevertheless, considerably harder than the capping. The latter is siliceous, seldom containing any clay or other binding material, and breaks into fine particles so that it runs like sand if given the opportunity. Because of these facts, and because the ore is above the average grade of the mine ore, it has been mined by top slicing.

Haulage levels are opened up 150 ft. apart, vertically, with two sublevels between at 50-ft. intervals, to facilitate the putting up of ore chutes. These sublevels are used during slicing for distributing air in the ventilation system. On the haulage level the drifts are spaced on 50-ft. centers, and raises along these drifts are also spaced on 50-ft. centers, except the incline raises as hereafter noted. The raises are cribbed where necessary. Where the wear will be excessive, 1/4-in. iron plates are spiked to the top of every third set of cribbing, for its protection.

When top slicing was first used, an attempt was made to carry a slicing face from fifty to several hundred feet long. Timber and other supplies were brought in through long drifts from an auxiliary shaft. Great difficulty was experienced in keeping these drifts open, the side pressure breaking the posts and the top weight breaking both caps and posts. Furthermore, the men could not work efficiently while these drifts were being repaired. The slicing faces advanced irregularly, due to varying conditions, and in many ways the results were not all that could be desired. It was then decided to divide the slicing area into blocks 200 ft. square and this was later changed to 250 ft. square.

At the center of each block a two-compartment raise is put up as a supply raise, the compartments being 2 ft. 6 in. and 4 ft. by 4 ft. 4 in., the smaller being used as a manway. Station sets of 12 by 12 timber with

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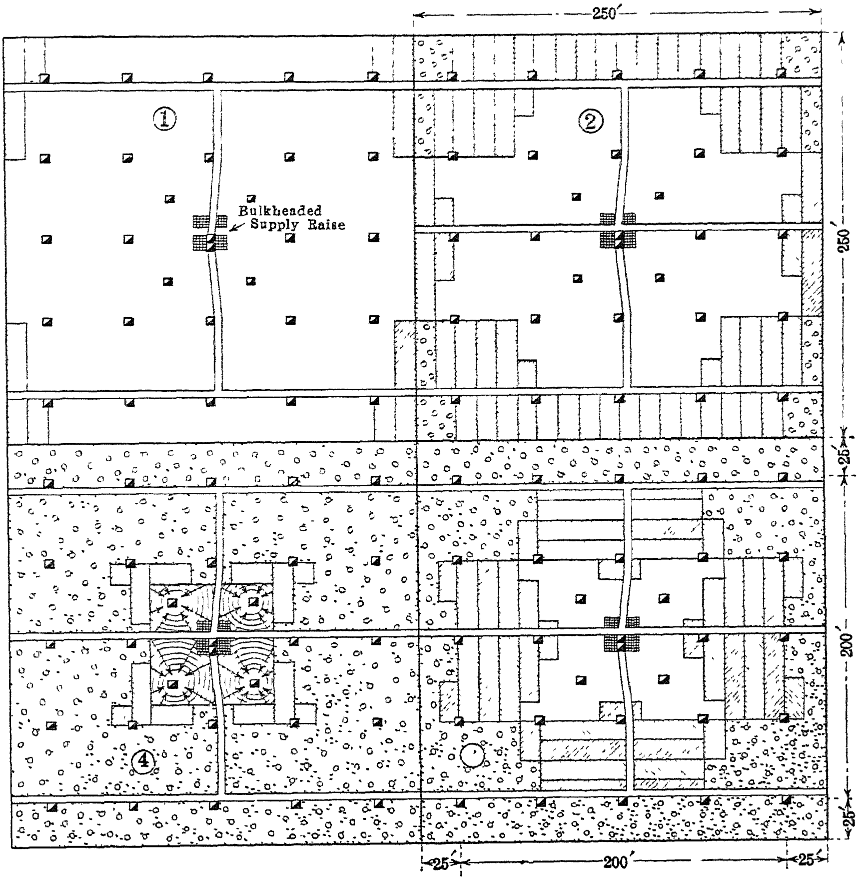
9- or 10-ft. posts are put in and an Ingersoll stretcher bar air hoist is mounted convenient to the larger raise compartment to use in hoisting the timber, steel, etc. Four bulkheads, built solidly of blocks of square timber, are put in as shown in Fig. 1. Two of these are 7 by 11 ft. in size, and the other two 7 by 7 ft.

Two drifts, usually untimbered at first, are run out 100 ft. on the long axis of the supply raise. At the end of each of these and at right angles



Figs. 1 to 4.—Plan Showing Progressive Steps in Block Method of Top-Slicing.

to them, two drifts are run 125 ft. to the block limits. When these limits are reached, slices are started toward the corners of the block. These slices are timbered either with single sets consisting of two 8-ft. posts and a 12-ft. cap, or with a double set consisting of three 8-ft. posts and two 7-ft. caps, depending upon the ground. The ore is taken to the floor above.

As soon as the first slices have advanced a few feet, second and third slices are started, and also first slices toward the centers of the block limits. Every man who can work to advantage is put on and the ore mined with the greatest possible speed.

Where there is a sufficient mat of old timbers in the back to obviate the danger of the capping running there is no permanent floor laid, the planks used in shoveling and wheeling the ore being taken up later. Where there is no mat or it is not sufficient, a floor of 2-in. plank spiked to 2 by 10 sills is laid. Formerly 5 by 10 and 4 by 8 sills were used, but it was found that after being subjected to the pressure and heat of a completed stope, a 5 by 10 or 4 by 8 sill seemed to have practically no more strength than a 2 by 10.

As soon as the timbers in the slices show signs of taking weight, bulkheads are built of old timbers, obtained either from the mat in the back, or from repair work in other parts of the mine. As soon as possible, the posts are drilled and the slices shot down.

By the time slicing has started, four drifts have been run to the centers of the sides of the blocks as shown in Fig. 2 and slicing is also done from these. Working as intensively as possible, all ore except the four central pillars is quickly mined out as shown in the series of illustrations. As about the last of this ore is being taken, crosscuts are driven to incline raises, put up to about the center of the four central pillars, as shown in Fig. 3, and slicing continues, working from the outside of the remaining ore to the supply-raise bulkheads first put up as shown in Fig. 1, thus completing the stope. By this time, these bulkheads, which were 10 ft. high when put in, have squeezed to from 4 to 6 ft. in height. Upon completion, the stope is shot down and another may then be started below, though it is best to let the ground settle for a few weeks.

At first thought the criticism suggests itself that with such weight the mining method used intensifies this weight as the block approaches completion. But experience has shown that, as a rule, the maximum weight is taken by the timbers at about the time the outside pillars are completed, and as mining progresses a larger proportion of the weight is taken by the bulkheads in the outside slices. At no time does the weight on the remaining ore and the slices still necessarily open get beyond control. These central pillars constitute our cheapest ore, not only because of the pillar raises but because the ground has been fractured by the weight, and lifters are the only holes necessary to break the face.

All drilling is done with plugger machines, using a water spray attached to a 5-gal. can. No cars are used in the slices, all ore being shoveled directly into the chutes or wheeled in barrows. Round timber is used almost exclusively in the stopes, because of its superior strength.

Ten feet has been taken as the standard height of a slice. If a greater

height is taken the ore sloughs off the top of the slicing face faster than it can be mucked out and the bottom shot, thus shortly caving the slice. It is possible that later, when a good mat has been formed, sublevel caving may be used, but so far, where tried, it has not been successful.

The ventilation of these slice blocks is very important because, without it, the heat coming out of the mat is excessive and prevents efficient work. It is accomplished by connecting one of the sublevels below the slicing floor with the discharge end of a 60,000-cu. ft. fan. Openings are maintained from the sublevel to raises through which it is desired to force air.

As a result of the change to the block method of slicing, and to using forced ventilation, the production per mucker-shift has been raised from 9 to 20 tons, the production per man-shift from 5 to 10 tons.

While the details of this method have been fully worked out and a large tonnage of ore has already been extracted by its use, the work to date has to a large extent been preparatory to systematic work for lower lifts. It is not possible to give representative costs, but the following is an estimate of what is expected:

Mining Costs per Ton

Preliminary development	$0.035	
Haulage development	0.025	
Other development (raises, etc.)	0.080	
Total development		$0.140
Stope Costs:		
Miners, at $3.75	0.080	
Muckers, at $3.75	0.160	
Drills	0.030	
Explosives	0.040	
Timbering, labor	0.070	
Timbering, supplies	0.130	
General—bosses, nippers, etc	0.040	
Total stoping		0.550
Haulage		0.055
Hoisting		0.04
Pumping		0.005
General underground		0.025
Ventilation		0.015
Engineering and sampling		0.016
Underground lighting		0.004
Mine surface		0.030
Total		$0.880

Discussion

The Chairman (P. G. Beckett, Globe, Ariz.).—The mining of large orebodies has in the last few years been such a big factor in the copper

output of this State, and, in fact, of the whole country, I feel that we should have some valuable discussion this evening on the various mining methods described in the papers which are to be read; and I hope you will start a discussion on Mr. Deane's paper.

J. A. EDE, La Salle, Ill.—I would like to ask whether, the timbers being left in, they have any fires owing to that fact?

E. G. DEANE.—No, we do not. The timbers in the mat seem to lose life. I cannot tell you exactly what change takes place, but they do not burn readily. There is undoubtedly a chemical change in them—a change that makes them lose all strength. I spoke of the strength of the sills. When we first started, we put in 4 by 8 sills, but those sills had no more strength in 6 months than a 2-in. plank would have.

J. P. HODGSON, Bisbee, Ariz.—I would like to ask Mr. Deane if the central supply raise in the center of the block has been satisfactory; and also whether in extracting the ore they put small raises up so that wheelbarrows are not used in the stope?

E. G. DEANE.—The supply raises have been very satisfactory. We do use wheelbarrows, as our raises are spaced 50-ft. centers; it may be that later it will pay us to put up small inclined raises. But just above the 420-level, where we have been doing our slicing, it has not paid us to do this.

J. P. HODGSON.—I would like to ask another question, and that is this: From your experience in top slicing, where have you found that the overburden decreases in weight? At about how many floors downward in course of extraction?

E. G. DEANE.—We have not gotten down that far. We have taken about ten slices in some parts of the mine, and have as much weight as ever. In other parts we are slicing immediately under the capping at the present time.

The Antecedent Mineral Discovery Requirement

BY E. D. GARDNER, M. E., MISSOULA, MONT.

(Arizona Meeting, September, 1916)

APPARENTLY the widespread agitation for the codification of our mining laws has had its effect, and it is quite possible that Congress will take up the question during this present session. The greatest objection to our present statutes pertaining to metalliferous deposits seems to be directed against the law of the apex. Those who are not in favor of changing this law are very far in the minority and they apparently realize that it is useless to urge its retention.

Next in importance to the repeal of the law of the apex, seems to be a widespread advocacy for the repeal of the law requiring discovery of mineral before a valid location of a mining claim can be made. To a lesser extent there appears to be a desire to have the law changed so as to allow patent for claims without mineral discoveries.

The purpose of this article is to review the arguments in favor of the proposed change of the discovery requirement, to show some of the evils that may be expected to result therefrom, and to consider whether a system may be devised to correct the undesirable features of the present law in this respect and at the same time not expose the mining fraternity and the general public to the evils which would result from the repeal of this fundamental requirement.

Under the law as at present framed, the validity of a mining location is dependent upon the fact of discovery of mineral within the boundaries of the land claimed.

"* * * but no location of a mining claim shall be made until the discovery of the vein or lode within the limits of the claim located" (Sec. 2,320, U. S. Rev. Stat.).

All authorities agree that discovery is the source of the miners' title. Lindley, in his valuable work on Mines (3d Ed., Sec. 335), lays down the rule in part as follows:

"Discovery is the initial fact. Without that no right can be acquired. * * * Such discovery must precede the location, or be in advance of intervening rights. The proof of recording and marking a claim will not authorize the court to presume a discovery."

As is well stated by the U. S. Supreme Court in Erhard v. Boaro (113 U. S. 536):

"A mere posting of a notice on a ridge of rocks cropping out of the earth or on other ground, that the poster has located thereon a mining claim, without any discovery or knowledge on his part of the existence of metal there, or in its immediate vicinity, would be justly treated as a mere speculative proceeding, and would not of itself initiate any right. There must be something beyond a mere guess on the part of the miner to authorize him to make a location which will exclude others from the ground, such as the discovery of the presence of the precious minerals in it, or in such proximity to it as to justify a reasonable belief in their existence."

Accordingly, if the miner has made a discovery and otherwise conformed to the requirements of the law, his possession of the claim located is exclusive, against all the world, even, in all probability, against the owner of the land, the Government, except on areas which Congress by special legislation has set apart and defined for paramount purposes, such as National Parks. He has an interest in the land which descends to his heirs, which he can dispose of by will, by deed, or other contract, and all that is required of him after location, until patent is secured from the Government, is to keep his claim alive by performance of the annual assessment work thereon.

In cases where two private locators claim the same ground, the one first making a discovery of mineral would probably get the area, irrespective of the relative time of beginning work or of posting notices. However, such cases are extremely rare, considering the total number of claims located each year. Strictly speaking, a prospector cannot make a valid location before he has discovered mineral, but he has the rights of possession, except against the United States; and, while there is a conflict in the decisions on this point, in many cases the rule appears to be that as long as he is working the ground and is in possession the law will protect him, and when a discovery is made his location is good.

In any attempt to remedy defects in the present system of Federal mining laws, the underlying intent and purpose of existing legislation on the subject should not be lost from view. As was well said by the Assistant Secretary of the Interior in Cataract Gold Mining Co. *et al.* (43 L. D., 248):

"The intent of the general mining laws was to encourage and promote the development of the mining resources of the United States."

Proposed amendments of existing law, not based on this broad policy but having in view the exploitation of the public domain in the interests of mere individuals without corresponding benefit to the people as a whole, should meet the condemnation of every honest and right-thinking man. Keeping in view, therefore, this beneficent purpose of the present law, let us consider whether the proposed elimination of the antecedent-discovery requirement from the law will operate to give greater effect to such purpose.

It is contended that, if the requirement of an antecedent discovery

of mineral within the lines of the claim located were dispensed with, the result would be to hasten the development of the mineral resources of the public domain and so increase the wealth of the nation; furthermore, that the present discovery requirement is an unnecessary hardship in a great many cases, and should be dispensed with in justice to those who are honestly endeavoring to develop the mineral resources of the public domain.

It is urged that, if the antecedent-discovery requirement were abolished, the one first staking out a piece of ground would be protected in his possession of it even if a later locator on the same ground were the first to discover mineral.

This raises the question, to whom should the desired reward (viz., exclusive occupation of the ground) be given? Shall it be to him who is diligent in staking out his claim, or to him who is not merely diligent in that respect but, not content with this, pursues diligence to the point of discovering the hidden mineral wealth? Which of the two is more deserving of reward? Which of the two is bending his efforts toward the consummation of the policy of the law? Certainly the one discovering mineral has performed a very essential act in the development of a mine.

In some mining districts where outcrops are few, extensive and expensive development work is necessary before valuable mineral is found. Many long and costly exploratory tunnels have been driven and shafts sunk to develop tracts of land on which occur no surface indications of veins or mineral but which were favorably situated in mineral belts. Justification for prosecution of development work under such conditions depends to a great extent on a careful study of mineral showing on adjacent lands. The successful prosecution of such exploratory work is dependent on the raising of sufficient capital for the enterprise. The raising of capital, a difficult undertaking in connection with almost any project so essentially speculative in character as that of exploring for precious metals, is made more difficult if the promoters cannot show a fee-simple title to the property to be developed. Hence, the abolishment of the discovery requirement would have the effect of overcoming this one obstacle to the development of our mineral resources.

When valuable orebodies are found on unpatented claims, there is always the menace of litigation to prove the ownership of the ground. After a strike has been made, it often happens that abandoned locations, which at one time covered the land, are resurrected or the boundary lines of prior adjoining claims are moved in such a manner as to take in the more valuable ground. The indefinite manner of describing claims in location notices, and the great number of old illegible mining corners usually scattered over the landscape in mining districts, increase the difficulty of successfully defeating such contests. The richer the ore-

bodies found in these cases, the stronger the efforts by contesting claimants to obtain the ground.

The danger of contests on unpatented ground after a strike has been made will be augmented if valid locations can be made without a discovery of mineral. Contests initiated by the locators of abandoned prior claims would be more likely to succeed if these claims had had a legal existence.

It may be well to say here, however, that it must not be taken for granted that all contests are not made in good faith. Locators of adjoining ground quite often do not know the position of each other's lines, and conflicts occur, therefore, when mineral survey is made for one or both of the claims. A contestant is usually honest in his belief in his ownership of the disputed area. Private contests against the issuance of patent for mining claims are being continually initiated by claimants for all or part of the ground.

Sometimes a locator tries to get part of another's prior location by getting his conflicting claim patented. I know of instances where valuable ground which was being held under location has been patented by a subsequent locator before the owner of the prior claim knew what was going on.

If we shut our eyes to actual conditions and assumed that all locators of mining claims act in perfect good faith in staking out ground with intent to develop it for its mineral contents, probably there would be no necessity for the antecedent-discovery requirement. But mining locations, even now, are made to cover a multitude of frauds against the public-land laws. Title to valuable water-power sites, immense quantities of timber and valuable town sites have been secured under the mining laws. How much greater and more frequent would be these frauds if even a perfunctory discovery of mineral were not required to validate the locations. By far the greater number of mining claims desired for purposes other than mining have had no discovery of mineral.

Care should be taken that the interests of all the people are not sacrificed in our consideration of the welfare of the prospectors. The discovery requirement is designed, and operates, as a check on the disposal of lands of the people under the mining laws for purposes foreign to the intent of the law, viz., the speedy and bona fide development of the mineral resources of the public domain.

If the validity of the location is to be dependent merely on taking possession of the ground without the antecedent discovery of mineral, what is to prevent the staking of claims near pay ground with intent to hold them for speculation and hold-up purposes, and not for bona fide development of mineral value, to a much greater extent than at present? What is to prevent irresponsible individuals from staking claims which can be held indefinitely by perfunctory assessment work, thereby tying up large areas of valuable ground for the purpose of levying tribute on

those desiring to obtain the ground for mining or other purposes? Surely, to abolish the antecedent-discovery requirement is to open the door to fraud and deceit incalculable, and will tend to block actual development of large areas for many years to come.

The abandonment of the mineral-discovery requirement would naturally facilitate the patenting of mining claims; but, however desirable it may be in some cases to expedite the granting of patents, I believe easy patenting, on the whole, will not benefit the mining industry, but will retard it.

Most mining men know of many promising claims and prospects for which patents have been obtained which have been, to all intents and purposes, abandoned. A large proportion of the applications for patent comes from owners who are tired of doing the annual assessment work but still desire to hold the ground. Of such cases, only a small number of claims are ever worked again.

Nearly 10,000 mineral surveys for groups of claims, from one to thirty or more each, have been made in the State of Montana alone, and this number does not include placer claims that have been taken up by legal subdivisions. While patent does not necessarily follow a mineral survey, it does so in nearly all cases, and of the patented claims only a small number have been worked since final certificates were issued. Outside of the Butte district, I will venture to say that less than 5 per cent. of the patented claims in Montana are now being worked, when metals are higher than at any time in the last 50 years.

In such active districts as Butte and the Coeur d'Alenes, where large capital is required in developing properties, the patenting of claims, on the whole, does not discourage new operations, but perhaps helps them. In cases, however, where one dominant company patents the whole surrounding territory, independent operators who would perhaps be willing to take a chance on developing some of this ground, if open, are thereby kept out.

The patenting of the ground seriously handicaps the chances of small camps and new districts for becoming important producers. Numerous cases can be pointed out where the patenting of the claims has seriously retarded the development of camps, or altogether stopped it.

The majority of those who have written about the desirability of amending the present mining laws have done so from the standpoint of the mining operator or prospector. None of the writers on the subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some

people would appear to believe. Individual claims on it are allowed only by virtue of the laws of Congress, which prescribe the performance of certain definite requirements, and it does not belong to the first comer solely on his assertion of a claim to any part of it.

Many abuses have been perpetrated by unscrupulous locators of land under the mining laws. Mining claims have been, and are now, located to control springs, water-holes, range privileges, power and reservoir sites, town sites, rights-of-way, summer-residence sites, timber and agricultural land, natural curiosities, and other surface values. While, at the present time, land more valuable for other purposes than for mining cannot be patented under the mining laws unless mineral has been found and the required expenditure made, in the past, before the Government inspected the ground before issuing patent, valuable areas were patented when the ground was in no respect mineral. At the present time, in some parts of the country, public business is seriously interfered with by unscrupulous locators of ground under the mining laws. While the Government is now embarrassed by these hold-up locations (the mining industry at large has no idea how great the number is), the main expense is the examination of such claims and the delay caused by them. If, by amending the present laws, valid locations can be made anywhere on Government land without mineral discovery, no Government or other activity on public land will be safe from extortion and blackmail. Conditions are bad enough now. For instance, in some localities, as soon as a body of Government timber is advertised for sale, the timbered area and ground over which the logs are to be moved to market is promptly located as mining claims, or old claims are revived, and heavy demands made for the use of the surface of the ground. If, by law, claims of this sort can be made valid, it will give unscrupulous individuals a monopoly of the surface and prevent legitimate business over large areas. The owner of a single valid mining claim crossing a right-of-way of a logging road could exact a sum, for the privilege of crossing his ground, large enough to take all the profit from the logging operations. As the cost of operation is directly related to what an operator can pay for the timber, the Government has a direct interest in all such cases. It has been repeatedly held by the courts that a valid location holds against the Government, even if abandoned for years. I know of areas from which the Government was contemplating selling the timber and on which there were as high as 100 abandoned mining locations. In such cases, if these locations had been valid without a mineral discovery, the locators on reasserting their rights to the ground as soon as they found the timber was valuable could have kept the Government from selling it. There is hardly a stream course or canyon in many parts of the West that has not been at some time plastered with locations.

Even if valid locations could be made without a mineral discovery

only on lands classified as mineral, it would still affect the Government, for there are bodies of white pine, which are advertised for sale from time to time, on lands in National Forests which have been classified as mineral but not covered by mining locations.

It has been said that one must actually make a discovery of mineral on land within National Forests before he is allowed possession or allowed to prospect it. Such is not the case, however. A prospector or miner is in no way disturbed by Government agents in the enjoyment of his rights within National Forests. No examination is made of any ground located for mining purposes unless it interferes with the administration of the Forest. A claim may interfere with the administration of the Forest when it conflicts with areas occupied by individuals under what are known as Special Use Permits; when it is included within a tract from which the Government has sold the timber; or where the claim is so located as to control rights of way over which it is necessary to transport Forest products. Unless the claims are actually interfering with the administration of the Forests, no examinations are undertaken until applications for patent are made. The proportion of mining claims examined prior to application for patent is very small, probably less than 1 per cent. of the total number located. There are thousands of mineral locations within the National Forests of which no examination has been made by the Forest Service, and in all probability never will be except in cases where application for patent is made.

Mining locations within the National Forests which, after an examination by a competent man, are shown to be clearly invalid, may be disregarded as far as the surface is concerned, but no action is taken to dispossess the locator of the ground and no objection whatever is made to his development of the mineral possibilities. In fact, the Assistant Secretary of the Interior has decided (Nichols-Smith case, unreported) that the Department has no jurisdiction over locations and has authority to cancel a claim only after an application for patent has been made. This decision, however, is now before the Secretary for consideration on motion for the exercise by him of his supervisory authority.

The fact that the Government has placed a large part of its domain within the National Forests shows that this land is valuable to the Government for the benefit of the people at large. This fact alone is an argument from the people's standpoint that the requirements for patenting Government land by individuals, at least within the Forests, should not be made easier.

I believe that, while sometimes the law requiring a discovery before a valid location can be made has worked hardships on prospectors, the wrong that could be worked under a law not requiring a discovery would be far greater.

Under the general mining laws, lands are not required to be chiefly

valuable for mineral before they can be entered. The Supreme Court of the United States has held in U. S. *vs.* Iron Silver Mining Co. (128 U. S. 673) that the fact that land may incidentally possess advantages other than its valuable mineral deposits will not preclude its disposition under the mining laws.

An act of Congress of Aug. 4, 1892 (24 Stat., 348) provided:

> "That any person authorized to enter lands under the mining laws of the United States may enter lands that are *chiefly* valuable for building stone under the provisions of the law in relation to placer mineral claims."

The point of difference between this act and the general mining law applicable to mineral deposits is that the Act of 1892 requires the land to be chiefly valuable for building stone.

Power or reservoir sites situated on land containing no evidence of mineral deposits have been entered as stone placers, but the applications for patent have been rejected by the General Land Office. If valid mineral locations could be made on such lands without a mineral discovery, the land could be patented as mining claims.

If patent were allowed for Government ground as mining claims without a mineral discovery there would be very few, if any, power projects free for development at the present time, for the ground would be validly held or patented as mining claims. Most of the undeveloped available power sites on public lands have been included in power withdrawals or National Forests. The ground is, therefore, not subject to entry except as mineral locations, and the only way to get a patent to such ground is under the mining laws. Nearly all power projects in the Northwest, where there is any possibility of early development, are on ground covered by mining locations. In some cases the mining claims are valid under the present law, but the majority have no mineral discovery. If the law were changed concerning mineral discoveries, there would be little new development of hydro-electric power without paying tribute to the kind of prospectors who had the foresight to locate mining claims covering the ground. Also Federal control, due to an interest in the site, could be circumvented by first obtaining patent as mining claims to all the land affected. In the Northwest there is a power project capable of developing 20,500 hp. One-half of the land affected by the project is within a National Forest, and therefore the Government under the present procedure exercises control over the development of the site. The power would be developed under a Special Use Permit issued by the Government, which requires certain conditions to be complied with. All of the public land has been located as placer mining claims, and if the claims were valid the Government under the present law would have no control whatever over the development of the power. In 1913, a permit was issued to a company to develop the power, but for some reason it was not

able to go ahead with the project. This company, however, acquired the mining locations, and if the claims could be patented the owners could prevent anyone else from developing the power site. Flour gold is found in gravel on all of these claims, but nowhere in paying quantities.

Another project with which I am familiar is capable of developing 331,000 hp. by using storage, or 178,000 hp. by utilizing only the minimum flow of the river. The lower 12 miles of this project, which is capable of developing 260,000 hp. by storage, or 140,000 hp. without, is entirely covered by mining locations. The area is within a mining district and would probably be classified as mineral, but only a few of the locations have mineral discoveries. One side of the river is within a power withdrawal, and the other within a National Forest. In this case, as in the other, a permit was granted to develop the project, but it has not been done. The permittee has acquired the mining claims and he controls the site in so far as his mining claims are valid. If all these mining claims were valid without mineral discoveries, they could be patented and the Government would lose control of the project.

On March 1, 1906, Windfield Doern and two others located the Eagle placer mining claims, alleged to be chiefly valuable for building stone, on the Stanislaus National Forest in California (41 L.D. 655–659). On Oct. 25, 1907, the claim was conveyed to the Stanislaus Electric Power Co., who filed application for patent on April 5, 1908. The entry was contested by the Government. Stone was used from the claim in the construction of a dam on the claim, from which water was conveyed to a plant 15 miles below, where power was generated. It was proposed to build a second plant on the claim and bring water in a conduit from above. The patent was denied by the General Land Office, as it was shown at a hearing that the stone had no value outside of the construction of the dam, and for that purpose had no special value. The land, however, had value as a power site.

On Jan. 2, 1907, H. V. Gates made application for patent under Act of Aug. 4, 1892, for the Excelsior placer, area 140 acres, in northern California. The patent was protested by the Northern California Power Co. At a hearing, it was disclosed that the ground contained a deposit of basalt that had only a local use for building stone, such as the construction of a power dam, and was not as suitable for that purpose as other common rock. The claim covered a reservoir and tunnel site, and the claimant had a permit from the Forest Service to use a part of the claim for power purposes. The General Land Office held that the claim was more valuable for power than for building stone, and the entry was canceled.

On a river in a western State is situated a power site capable of developing 2,490 hp. The dam site is covered by an unpatented lode claim, and above the falls are situated four patented lode claims along the river in

a row. Each of the four patented claims contains a valid mineral discovery, but no ore. Before proceeding to patent, the claimants received $5,000 from a power company for the privilege of flooding their ground.

Perhaps more fraudulent mining locations have been made to acquire valuable timber than for any other purpose on non-mineral land. If the mineral-discovery requirement is repealed, large timber steals which have been prevented in the past will be successful. Many applications for timbered mining claims have been canceled after the Government has shown that no mineral discoveries have been made. It would not be possible to defeat such cases if the proposed change of the law is made.

Timber on patented mining claims can be disposed of by the owner as he sees fit. Timber on valid unpatented mining claims on proven mineral land outside of National Forests can be cut for certain described purposes as approved by Congress by Act of June 3, 1878 (20 Stat., 88). Timber, however, on an unpatented mining claim within the National Forests cannot be cut for any purpose other than the development of the claim, except by Special Use Permit or purchase from the Government.

It has been a common practice in some localities to locate mining claims on non-mineral timbered areas to give color of title to the ground while cutting the timber. Numerous suits have been instituted by the Government to recover the value of timber cut from such invalid claims.

In the case of Anderson *vs.* United States (152 Fed., 89 to 91), Anderson, Baker and Sandlin located as placer claims 1,200 acres of non-mineral Government land in the Boise land district, Idaho, from which was cut timber worth $17,751.79. Suit was brought by the Government to recover the value of the timber, and it was successful.

In the case of Powers *vs.* United States (119 Fed., 562), the Government brought suit to recover the value of 668,000 ft. b. m. of timber, worth $7,241, which was cut from non-mineral Government ground. The ground was covered by mining locations made by a third party.

About 10 years ago, a company applied for patent for 3,636.99 acres of Government land as placer claims. The magnitude of the case caused examination to be made by the General Land Office, and it was found that on one claim there was a small worked-out placer deposit of gold. There was no possibility of gold occurring in paying quantities on the rest of the group, although fine colors could be found almost anywhere in the country. This land contained an average stand of 15,000 ft. b. m. of white pine and other varieties to the acre, or about a total of 60,000,000 ft. b. m. on the group. The claims were continuous, but excluded and almost surrounded a small area from which the timber had been burned. The entry was protested by the Government and at a hearing held to determine the validity of the group, the claimants defaulted, thereby admitting the truth of the charges. The entry was finally canceled on Jan. 28, 1910. At this time, unpatented fraudulent placer claims were

being held in this district which contained a stand of between 200 and 300 million feet of timber. During the summer of 1910, forest fires that destroyed the timber swept over this country, after which all of the timber-mining locations were abandoned.

In 1913, application was made for two groups of claims in Idaho. No mineral had been found on four of the claims of the group which contained a heavy stand of white pine worth $3,800. After a hearing, the entry was held for cancellation by the Commissioner of the General Land Office.

In 1912, application was made for two claims in the Coeur d'Alene district, Idaho. This entry was also held for cancellation by the Commissioner of the General Land Office, after a hearing. These claims were completely surrounded by patented placer claims, but were situated on a small mountain. The ground was covered by a stand of white pine having a stumpage value of $3,478. One of the claimants was interested in a saw mill situated near the claims.

Along in 1906, a mining company (long since defunct) did considerable development work on lode locations in western Montana, and located a large number of claims. The company failed to find any ore, and went out of existence. The first year after the company abandoned the ground, it was located by a man who did no more work on it. He relocated some of the claims for several years, but when no one else showed any desire to obtain the ground, he also abandoned it. In 1914, a lumber company secured timber on the river above these claims. After the lumber company cleared its right-of-way, the mineral locator came along, saw the situation and immediately reasserted his right to some of the ground that controlled the right-of-way. He demanded an excessive sum for the privilege of crossing his ground. His demands were refused and logging operations shut down for about a year. An examination of the mining claims was made by a mining engineer and it was found that the claims were invalid because no mineral had been discovered upon them. After a year's delay, the company constructed the logging road and commenced operations without any further trouble and without paying tribute to the locator. If these locations were valid without mineral discoveries, the logging operations could have been held up indefinitely unless the excessive demands of the locator were met. The claims are in a general mineral belt.

If valid locations of lode claims can be made in the future without a mineral discovery, many town sites are going to be held as mining claims. The following case is now before the Land Department for settlement. In 1909, a lode claim was located in such a manner as to take in all of the level portion of the canyon at the division point of a new railroad in Idaho. On June 27, 1913, application for patent was made. The section in which the claim is situated has been classified by the United

States Geological Survey as non-mineral, and the land is in the Northern Pacific grant. The railroad has a station and some dwelling houses on the ground and part of the claim is also claimed by the heirs of a homesteader. If the mineral locator can prove a valid discovery on the claim, he is entitled to the ground. The railroad began operating about the time the claim was located. Practically a whole town, with a population of several hundred, is on the mining claim, and there is no doubt that the ground is more valuable for town-site purposes than for anything else. A large diabase dike, in which can be found traces of copper, runs through the claim, but there is no indication that paying ore will ever be found upon it. Sufficient development work has been performed. If no discovery of mineral is going to be required, many similar cases may be expected in the future.

The repeal of the mineral-discovery requirement would open the doors for blackmail of all irrigation companies situated like the one in the following case, even when there is not a trace of mineral on the land. In 1910, an application for patent was made for a placer situated in a small park in western Montana. Previous to the mineral application, an application was made to the Secretary of the Interior for the use of the park for a storage-reservoir site, for which purpose the ground is well adapted. It was proposed to impound water during the winter months and high-water periods, to be used for reclaiming about 3,000 acres of arid land. The reservoir permit was approved by the Secretary, but construction has not been begun on account of the placer claim. The Commissioner of the General Land Office decided that sufficient mineral to constitute a discovery had not been made upon the claim and it is now before the Secretary of the Interior on appeal. The park is an old lake bottom and contains sand, clay and gravel to an undetermined depth. This material all contains minute quantities of gold, but in no place, as far as developed, in paying quantities. No serious effort has been made to mine the gravel, and for the last 20 years the ground has been used by the claimant as a ranch. If it is finally decided that the claim contains sufficient mineral for a discovery, the mineral claimant will be legally entitled to the land, and, before the irrigation system can be commenced, the land will either have to be obtained from him by purchase or by condemnation proceedings.

Throughout the country, attempts have been made to obtain possession of numerous springs, mineral or otherwise, by taking up the ground as mining claims. On Aug. 26, 1886, the Margaret Mining Co. made application for the Gray Eagle Lode mining claim (11 L. D., p. 563) in King County, Washington. This claim embraced some springs known as the Hot Springs of Green River which were supposed to have great medicinal value. The entry was canceled by the Land Office because it was shown at a hearing that the land had no value for mineral but was

used for a health resort, and that the mining company was formed to acquire title to the Hot Springs.

By the Act of Jan. 31, 1901 (31 Stat., 745) Congress provided that "all unoccupied public lands of the United States containing salt springs or deposits of salt in any form and chiefly valuable therefor are hereby declared to be subject to location and purchase under the provisions of the law relating to placer mining claims." Under this Act many mineral springs having no value whatever for salt, but having supposedly medicinal properties, have been located as placer claims.

On March 20, 1905, Henry Lovely made application for patent for the Lovely placer claim in the Juneau land district in Alaska. It was shown at a hearing that the ground was used as a health resort and water from two springs was conducted to bath houses, the cost of this being applied as mining improvements. It was also shown that the ground was not mineral or saline in character. The claim was canceled Feb. 13, 1907, by the General Land Office (25 L. D., 426).

By decision of April 14, 1913, the Secretary of the Interior held that the Salt Creek placer claims were invalid (21 L. D., 745). These claims were located in February, 1911, under the Act of Jan. 31, 1901, so as to cover a group of hot springs whose waters carried in solution common salt, Epsom salts and Glaubers salts and other chemical substances. As compared to sea water, the waters of these springs were shown to contain about one-eighth as much common salt. It was held by the Secretary of the Interior, from facts determined at a hearing, that the ground had no value for the manufacture of common salt, but possessed distinct curative properties.

In central Montana is situated a mineral spring known as the Appollonaris. The ground at the small side gulch on which the spring is situated was located as a stone placer and the adjoining ground on the main creek as gold placers. No discovery of gold had been made on the gold placers, and the limestone on the stone placer was not valuable on account of its poor quality and the distance from any market or railroad. This spring has a local reputation for medicinal properties and Special Use Permits for erecting bottling works and hotel buildings on the area was desired by several people. The development of the spring was held up for 5 years, but finally the men holding the mining claims decided that they could not patent the ground and abandoned it. The spring is now being developed.

The abrogation of the mineral-discovery requirement would greatly facilitate such fraudulent attempts to obtain ground desired for purposes other than mining.

At numerous points desirable for summer residences on the shore of Lake Pend Oreille, mining locations have been posted, and in many cases there has been a pretense of doing the annual assessment work. In a

few cases, application for patent has been made. On some of these claims sufficient mineral has been found to constitute a mineral discovery, but in the majority of cases no mineral has been found.

The Frances Placer, consisting of 20 acres on a bar on the Clearwater River in central Idaho, was applied for under the mining laws. The claimant had farmed the land for at least 10 years previous to making application. The mineral entry was contested and canceled in 1912, and the land was later taken as a homestead.

Mining claims have been patented, and others are being held as locations, on favorably located scenic points along the rim of the Grand Canyon of the Colorado. Two lode claims also have been patented which cross the Bright Angel Trail. Tourists who have gone down this trail and paid their dollar each to the owner of the lode claims can realize his farsightedness in finding his mine just in this particular location. It is of interest to note that no mineral commercially valuable has been found on any of these claims.

On the public domain there are numerous conflicts of mineral claims with homesteads. In all contests, the burden of proof is on the homesteader, as he has to prove the land non-mineral in character. If valid locations can be made without the necessity of a mineral discovery, the homesteader's lot in mountain valleys is going to be still harder to bear, and the good Lord knoweth it is bad enough now.

If the mineral-discovery requirements for the location of mining claims is abrogated, no form of activity can be safely undertaken on unpatented land, and it is difficult to anticipate the many abuses that can be perpetrated.

The uncertainty of ultimate possession of unpatented land in mineral belts could be remedied, I believe, by making provision by law for clearing title to the ground against all except the United States at any time, as is now done at the time of application for patent. If, after development, mineral is found, the ground could then be patented.

Perhaps the method now used in connection with homesteads on the public domain could be adapted to the making of mineral entries. In the case of homesteads on surveyed land, an applicant, at the time of going on the land, makes a filing in the U. S. Land Office, corresponding to the posting of a location notice on a mining claim, and, if no adverse claims are proven and the land is non-mineral, his entry is allowed. This excludes all other private claims to the ground, and, after he has complied with certain prescribed regulations, patent is issued to the homesteader.

If all men were honest, or this country were a Utopia, it would be perfectly feasible to repeal the law requiring the discovery of mineral, but, unfortunately, such is not the case. Under the present law some operators may have been unjustly obliged to defend their rights, but it has been found that laws made to restrain the unjust usually cause suffer-

ing to some of the just. The repeal of this law would afford relief to some who are acting in good faith, but at the same time it would let down the bars, allowing an unlimited amount of fraud, and fostering blackmail. That the wrong application, if possible, would be made of any new mining laws, should be expected from the many frauds that have been perpetrated under the present laws. By far the greater number of the mining claims that have been desired for purposes other than mining have had no discoveries of mineral. I believe that by suitable laws it is quite possible to ameliorate the one condition without aggravating the other.

Discussion

Victor G. Hills, Denver, Colo. (communication to the Secretary*).—If anyone advocates the abolition of the antecedent mineral discovery requirement for the purpose of making it easier to secure title to public land, I hasten to say that I am not of that number. I am in hearty accord with all that Mr. Gardner has to say in regard to easy patenting as a handicap to small camps, and that in general it acts to retard the mining industry. And the following sentences I want to repeat in order to emphasize and endorse them, "None of the writers on this subject seem to have taken into consideration the effect amending the law pertaining to mineral discovery will have upon the Government's administration of the public land. It must be borne in mind that this land belongs absolutely to the people of the United States as a whole, and their interest must be considered. It is not a no-man's land, as some people would appear to believe." However, I believe that the discovery prerequisite should be abolished because it fails to effect the protection for which it was designed, and I look for other provisions that will better serve the purpose.

We propose to abolish the law of apex not because the theory is objectionable but because the question of physical fact gives rise to never ending litigation. Now the design of antecedent mineral discovery is good; the court rulings are unobjectionable; but when it comes to the physical fact this requirement is second only to the apex law as a healthy litigation breeder. The only circumstance that has prevented the prediscovery lawsuits from becoming as famous and as expensive as the apex suits is that these cases are fought prior to patenting when usually there are not the valuable orebodies proven to stimulate greater expense. The question of discovery can not be raised after patent. In the pre-patent days disputes and conflicts are settled largely by compromise and the public is cheated right and left. In the course of my observation of claim disputes, and it has been long and extensive, the only question seriously raised, as a rule, is whether the contestant's "discovery shaft"

* Received Nov. 3, 1916.

has rock in place or is only in wash, and even the question of wash has been successfully disputed and a good assay allowed to save the day. If the discovery shaft has reached rock in place it is too well known that any crack in the rock, any oxidized seam, or any pegmatite streak in the granite, will "hold" if the question comes to a jury, and the other claimants usually throw up their hands to the stake with the oldest date. If there is no conflicting claimant the location is recognized and patented without so much as a thought of the public interest. Thus the discovery requirement, by being depended upon to safeguard against the frauds of securing springs, water holes, range privileges, power and reservoir sites, rights-of-way, summer residence sites, timber land, natural curiosities, etc., becomes a lever to assist in fixing such frauds. A crevice in the rock can be found almost anywhere, and there is nothing with which to contest a mineral title. When, as occasionally happens, two parties race for discovery on the same piece of ground, it is not ore they look for but any crevice in the rock which will serve as an excuse for a "vein." Quoting from Mr. Gardner's paper (p. 248), "The discovery requirement is designed, and operates, as a check on the disposal of land of the people under the mining laws for purposes foreign to the intent of the law, viz., the speedy and bona fide development of the mineral resources of the public domain." The design is all right, but the operation is a shameful failure. Indeed, in the next to the last sentence of the article we read (p. 259), "By far the greater number of mining claims that have been desired for purposes other than mining have had no discovery of mineral." It is not the use, but the abuse of the discovery requirement which renders it desirable to discard it. We should remember the adage that a law which can not be enforced is worse than none.

Let no one rise and say that we should have some one detailed from the U. S. Geological Survey to examine every discovery, at least prior to patent application. That would make only a little more interesting sport than heretofore. The geologist would come into court and say that there was not sufficient showing to constitute a mineral discovery, and his report would be true and just; then would come the "prospector" with his assay, scraped out of a knife-blade seam (and taken in a sack which had been previously used in sampling high-grade ore); the next witness would say that there was "a sufficient showing to justify a miner in following the same with a reasonable expectation of finding pay ore" —that is standard court language which we all know by heart; the next witness would say that he once saw the famous .... mine, which has produced millions, when it looked no better than the one in dispute; then would come the jury and give the claim to the "poor prospector," every time!

Now the question will naturally arise, "What substitute have you to offer for the protection of the public domain?" In reply, I would say, that

in the first place the situation could scarcely be worse with the discovery antecedent removed. Next, the proposal to require recording a notice of annual assessment with vouchers and strict penalties has been so universally endorsed that we may regard it as certain to be included in any new law. This provision will entirely do away with the blackmailing scheme of resurrecting dead claims when some one else has proven the ground valuable. Also, since the only claims which can remain alive after one year are those on which the locator makes a bona fide expenditure, it will very largely do away with the fraudulent practice of holding ground for timber, springs and other surface values. There is an effective difference between an actual expenditure of $100 and an assessment which can be contracted for from $10 to $25. Further, I should favor at least $10 per acre for annual assessment. This change alone will do more to protect the public domain than the pre-discovery requirement has ever done. The fact that more is charged for mineral land than for any other class (and the preliminary expense of patenting is also greater) safeguards agricultural land, summer residence sites, etc. The proposal made by one writer,[1] that, when patenting, $100 per acre be charged when there is no discovery, is worthy of consideration; but I would rather be rid of the discovery question altogether.

The one great thing which would do away with all of our troubles on the discovery question, and also a lot of other mining-law troubles, is the divorce of surface and mineral titles. I frankly acknowledge the radical nature of such a proposal, and presume that a vote among the fraternity would show me in a sad minority, but I exercise the courage of my conviction. The famous 33 questions which have been put forth by the Mining and Metallurgical Society do not bring out this most vital question, perhaps because it was regarded as practically useless. However, several members in their discussion recur to this subject, and evidently this matter forces itself upon the mind of every thorough thinker on the subject.

The use of the surface and the extraction of minerals do not, except to a limited extent, naturally belong together, and any law which persists in keeping the two inseparable must be full of injustice and trouble breeding. Why not meet the main difficulty squarely? Grant the prospector the full and exclusive right to the minerals which he claims to be searching for, without discovery, together with all of the surface which he desires to actually use, but allow all other citizens the surface right-of-way for roads, trams, pipe lines, ditches, etc., and have the Government retain control of mineral springs, reservoir sites, and the like. Then all of the abuse found in the use of mining locations for hold-up purposes will disappear, and with it most of the urgency for pre-discovery.

[1] *Engineering and Mining Journal*, vol. 12, No. 16, ¶ C, p. 721 (Oct. 14, 1916).

Our mining companies, after all their ground is patented, frequently buy and sell surface without mineral rights and mineral titles without the surface. They simply meet a natural physical condition. Why should not the Government do the same? Let the patent applicant pay $5 per acre for the mineral rights and $5 per acre additional for whatever surface he requires. If there be a specially valuable tract of timber, beyond the needs for mining the property, a spring or a town site, let the Government inspector assess the value of the same and grant the patentee the choice of buying or eliminating such portions of the surface. Any such surface exclusions can be monumented at the time of the official survey and the Surveyor General will have a map of both surface and mineral titles with scarcely any additional expense.

Separating surface and mineral titles also disposes of the "Classification" question, which I do not consider a practical thing in any case.

I favor making annual assessment work at least $10 per acre and pre-patent work $50 or $100 per acre. This is no more than Colorado, with its 10-acre claims, has always paid, and such a provision would help as a safeguard in the absence of the antecedent discovery.

If the separation of surface and mineral titles can not be accomplished, why can not the Government at least retain control of certain surface privileges during the pre-patent period, and at patent application make an assessment of special surface values and require payment for the same in addition to the nominal per-acre charge?

Petrography of the Mount Morgan Mine, Queensland.

BY W. E. GABY,* B. S., M. A., BUTTE, MONT.

(Arizona Meeting, September, 1916)

INTRODUCTION

SINCE the time of their discovery, the genesis of the ores at Mount Morgan, and the nature of the changes which have affected the surrounding rocks, have been the subject of investigation and speculation by geologists and mining engineers. Developments at this mine, for a long period the greatest gold producer of the world, show that with depth the orebody grades into copper, of which metal it now furnishes a large output, exceeding in value the gold.

In earlier times, the purity of the gold and the first extremely heterogeneous character of the ore made it an interesting problem. Now, the probable depth of the copper enrichment, genesis, and true relations of the neighboring rocks furnish equally profitable material for investigation, and the present somewhat detailed description of these ores and rocks, from microscopic study, may prove of interest.

The mine occupies a highly silicified portion of a belt of quartz-porphyry, which is believed to be the oldest rock of the region. This belt of quartz-porphyry and allied rocks runs north and south between two areas of a later intrusive granite, as shown on the geologic map,[1] Fig. 1. Near the orebody the quartz-porphyry has been intruded by a dense basaltic mass called the "Old Basic." Cutting all three of these igneous masses are two sets of basaltic dikes, one running northwest-southeast, and the other, cross dikes, northeast-southwest. These dikes may be contemporaneous with, instead of later than, the ore. Their number has caused peculiar difficulties in extracting it, as the clay selvage of decomposed material affords no adhesion between dike and ore, causing great blocks to loosen and come down.

The investigation was based on specimens furnished by B. Magnus, former general manager of the mine, and notes on the areal geology by

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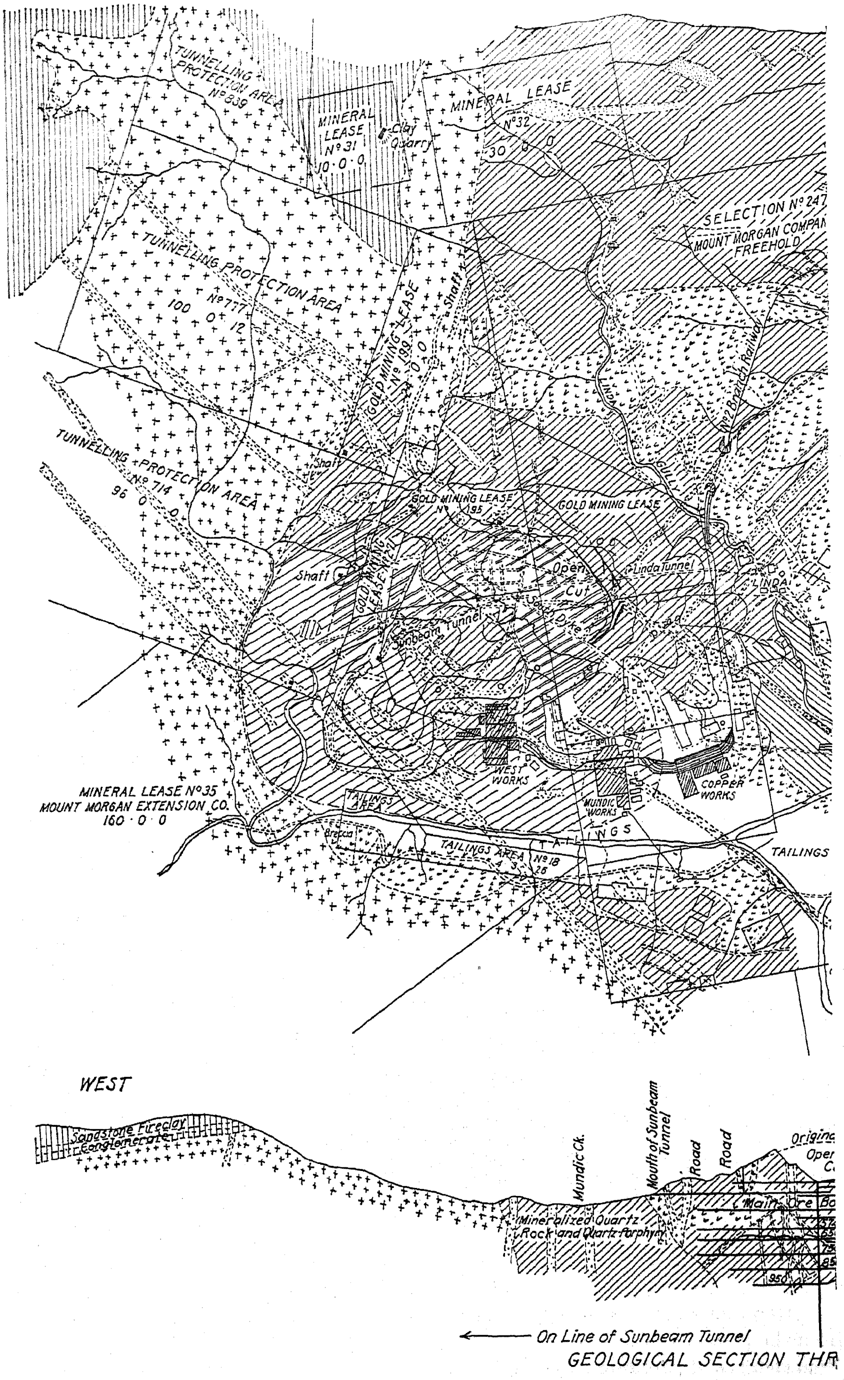
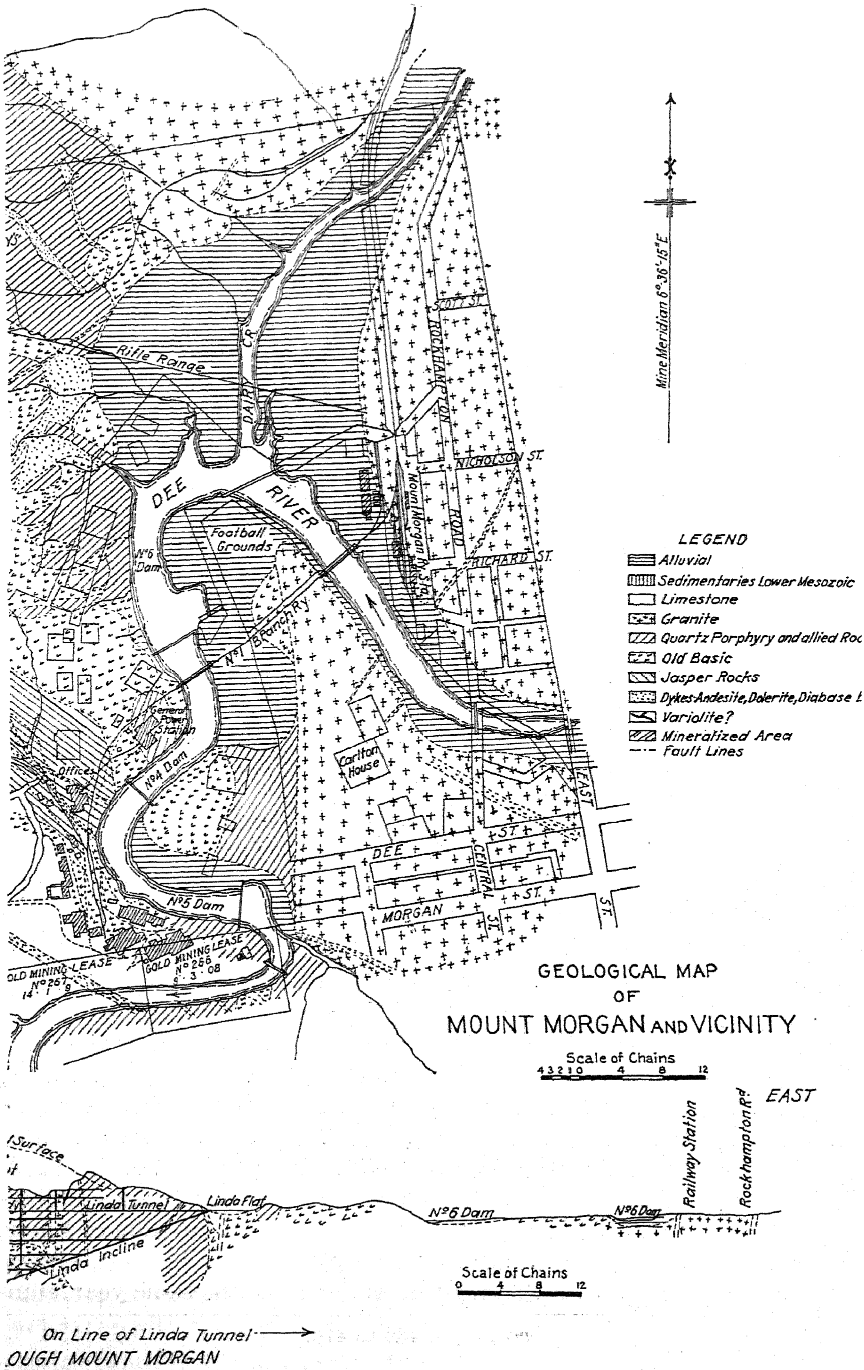
* Assistant Geologist, Anaconda Copper Mining Co.

[1] Reproduced by permission of the Australasian Institute of Mining Engineers from the plate accompanying the paper of J. M. Newman and G. F. Campbell Brown, *Transactions, Australasian Institute of Mining Engineers*, vol. 15, part 2 (1910).



TUNNELLING PROTECTION AREA Nº 839
MINERAL LEASE Nº 31 10·0·0
Clay Quarry
MINERAL LEASE Nº 32 30 0 0
SELECTION Nº 247
MOUNT MORGAN COMPAN
FREEHOLD
TUNNELLING PROTECTION AREA Nº 777 100 0 12
GOLD MINING LEASE Nº 199 24 0 0
Shaft
Nº 2 Branch Railway
TUNNELLING PROTECTION AREA Nº 714 96 0 0
GOLD MINING LEASE Nº 195
GOLD MINING LEASE
Open Cut
Linda Tunnel
LINDA
Sunbeam Tunnel
WEST WORKS
MUNDIC WORKS
COPPER WORKS
MINERAL LEASE Nº 35
MOUNT MORGAN EXTENSION CO.
160 · 0 · 0
TAILINGS AREA
TAILINGS AREA Nº 18 4 3 28
TAILINGS
TAILINGS
WEST
Sandstone Fireclay Conglomerate
Mundic Ck.
Mouth of Sunbeam Tunnel
Road
Road
Mineralized Quartz Rock and Quartz Porphyry
Main Ore Bo
On Line of Sunbeam Tunnel
GEOLOGICAL SECTION THR

Fig. 1.


Mine Meridian 6°-36'-15" E
LEGEND
Alluvial
Sedimentaries Lower Mesozoic
Limestone
Granite
Quartz Porphyry and allied Roc
Old Basic
Jasper Rocks
Dykes Andesite, Dolerite, Diabase
Variolite?
Mineralized Area
Fault Lines
GEOLOGICAL MAP
OF
MOUNT MORGAN AND VICINITY
Scale of Chains
4 3 2 1 0 4 8 12
Rifle Range
DAIRY CR.
DEE RIVER
Football Grounds
No 6 Dam
No 1 Branch Ry
General Power Station
Offices
No 4 Dam
No 5 Dam
Carlton House
SCOTT ST.
ROCKHAMPTON ROAD
NICHOLSON ST.
Mount Morgan Ry. Sta.
RICHARD ST.
DEE ST.
MORGAN ST.
CENTRAL ST.
EAST ST.
GOLD MINING LEASE No 266
OLD MINING LEASE No 267
EAST
Railway Station
Rockhampton Rd
Surface
Linda Tunnel
Linda Flat
No 6 Dam
Linda Incline
Scale of Chains
0 4 8 12
On Line of Linda Tunnel
OUGH MOUNT MORGAN

J. M. Newman and G. F. Campbell Brown.[2] Reference to the literature shows a wide divergence of opinion concerning its geology to have existed at different times.

Summary of Opinions on Genesis of Mount Morgan Orebody

The first examination was made by R. L. Jack, as geologist for the Queensland Government, in 1884, and he came to a conclusion, from the

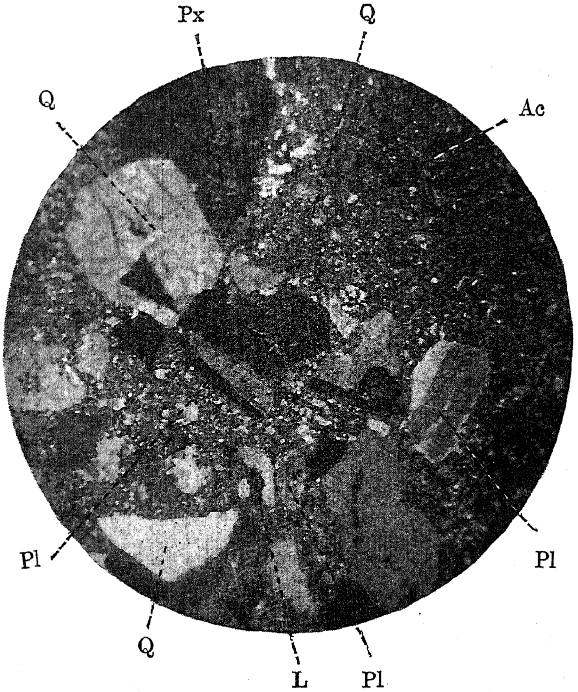


Fig. 2.—Quartz-Porphyry of the Composition of Quartz-Diorite or Quartz-Monzonite, with a Ground-Mass of Fine Granular Secondary Quartz, Chlorite, Leucoxene, and Hematite. Plagioclase and Quartz Make Up the Phenocrysts, and Patches of Pyroxene are Common. The Latter has been Partly Altered to Chlorite and Epidote. Crossed Nicols. × 24.

evidence then available, that nothing but a thermal spring in the open air, or geyser, could account for the formation of the orebody. In 1891, Walter H. Weed, of the United States Geological Survey, was asked by Mr. Jack to examine and compare a suite of specimens from the mine with the siliceous sinters of the hot spring region of Yellowstone Park. Weed came to a similar conclusion, citing the Steamboat Springs of Nevada, which have long been known to be surrounded by deposits of sinter, in fissures of which ore deposition is taking place.

T. A. Rickard, after an examination of the mine the same year, sum-

[2] *Loc. cit.*, pp. 439 to 470.

marized the various views held up to this time as follows: 1. That the deposit is one of a geyser (R. L. Jack, 1884). 2. That it is an auriferous zone traversed by a series of quartz veins of auriferous mundic (gold-bearing pyrite) (J. MacDonald Cameron, 1887). 3. That it is the decomposed cap of a large pyrite lode. (The view held by several local and other mining engineers at the time of his examination.)

Mr. Rickard dismissed the geyser theory as an altogether local appearance, based as it was on a small open cut which showed a fan-shaped

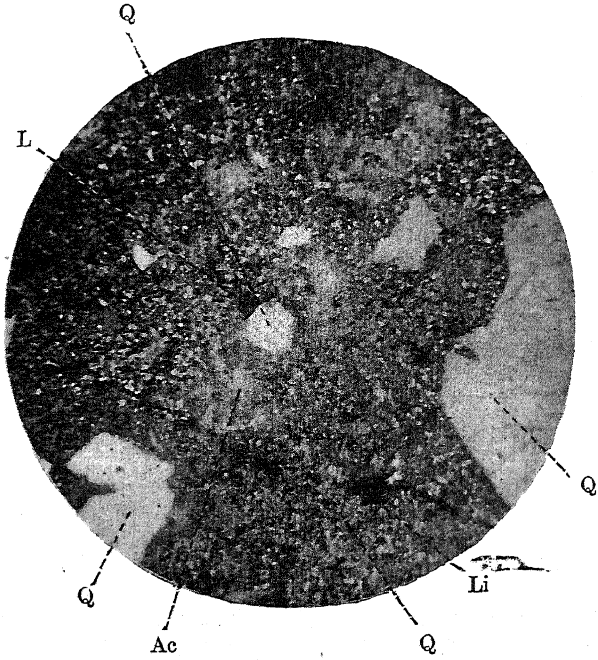


FIG. 3.—QUARTZ-PORPHYRY. THERE IS A STRONG DEVELOPMENT OF ACTINOLITE AMONG THE SMALL GRAINS OF SECONDARY QUARTZ WHICH MAKES UP THE GROUND-MASS, PROBABLY A RESULT OF PROXIMITY TO THE BASALTIC ROCKS. LEUCOXENE (WHITE AND OPAQUE, HENCE DARK) IS ABUNDANT. PHENOCRYSTS OF QUARTZ, AND GROUND-MASS LARGELY SECONDARY QUARTZ. LIMONITE, AS EVIDENCE OF WEATHERING, OCCURS DISSEMINATED AND ALONG FRACTURES. CROSSED NICOLS. × 24.

arrangement of ore, and his own explanation of Mount Morgan was, that it "represents an altered portion of shattered country rock, which, by reason of its crushed condition, was readily acted upon by mineral solutions, and that these solutions replaced the basic and feldspathic with acidic and quartzose material which was also gold-bearing."

The geyser and replacement theories were later reviewed by Dr. Frank D. Adams of McGill University with an interesting account of the discovery and subsequent history of the mine. But by 1899, the reaching of depths below the zone of oxidation revealed data which did not agree with some of these earlier conceptions.

Many different names and origins have also been assigned to the rocks of the district. Metamorphism was supposed by some to have transformed original sedimentary rocks into the present ones, which are, as is generally stated, difficult to distinguish from igneous rocks. The supposed graywacke, quartzites, and shales of sedimentary origin are now known to be igneous or derived from igneous rocks. There are no true sediments near the mine except those to the northwest of it, which overlie

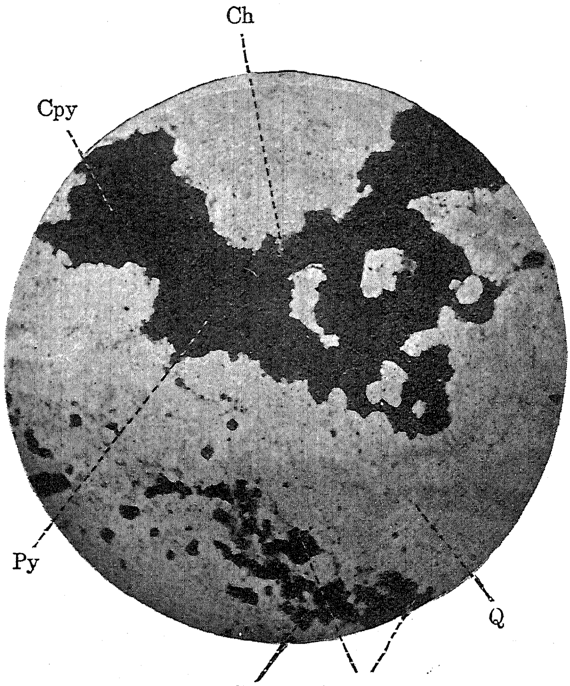


FIG. 4.—ORE, SHOWING PYRITE AND CHALCOPYRITE WITH CHLORITE IN GANGUE OF QUARTZ, THE CHLORITE WITHIN AND AROUND THE SULPHIDES, ALL THREE APPEARING TO OCCUPY THE SPACES AND CHANNELS BETWEEN THE QUARTZ GRAINS. SOME SMALL GREEN NEEDLE-LIKE CRYSTALS, NOT PLAINLY SEEN, ARE PROBABLY ACTINOLITE. PLANE LIGHT. $\times$ 24.

the very latest of the intrusive masses. These sediments are Lower Mesozoic in age and consist of conglomerates, fireclays, and sandstones, with a possible bed of volcanic ash or tuff. No specimen of the latter material was available, so this point could not be determined.

The notes by Newman and Brown, and by G. S. Hart, in the *Transactions* of the Australasian Institute of Mining Engineers, from whose maps and descriptions the field relations are taken, represent the most recent published work on the mine.

It is hoped the additional facts here set forth, from microscopic study, may be of assistance in solving the problem of the origin of its ores, as well as give additional information concerning the associated rocks.

In some cases the names of rocks have been changed to more nearly fit their mineralogy and textures as revealed in thin section, the names here given according with the American usage. Acknowledgments are due the authors of the above papers, to Mr. Magnus for specimens and notes, and to Dr. Charles P. Berkey of Columbia University for his kindly assistance in the laboratory.

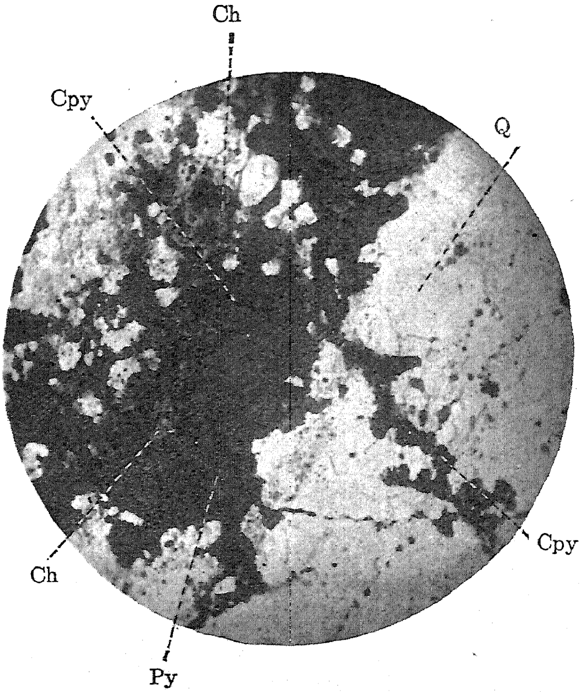


Fig. 5.—Ore, Showing Relation of Chlorite to the Sulphides, and Distribution of the Pyrite and Chalcopyrite with the Chlorite along the Veinlets and Spaces in the Quartz. Plane Light. × 24.

Description of Micrographs

The accompanying geologic map, Fig. 1, will make clear the general relationship of the Mount Morgan rocks, and with this in mind, the descriptions follow in the order of their relative geologic age.

Key to Minerals in Photomicrographs

Ac	= Actinolite	L	= Leucoxene
Ap	= Apatite	Li	= Limonite
Aug	= Augite	M	= Magnetite
C	= Carbonate	Pl	= Plagioclase
Ch	= Chlorite	Px	= Pyroxene
Cpy	= Chalcopyrite	Py	= Pyrite
Hb	= Hornblende	Q	= Quartz
Il	= Ilmenite		

Quartz-Porphyries and Ore

These rocks are grayish green and porphyritic with a fine-grained ground-mass and scattered irregular-shaped crystals of quartz. Two photomicrographs, Figs. 2 and 3, show them in thin section. Plagioclase is the predominant feldspar and ilmenite is the chief accessory mineral. Secondary development of quartz, actinolite, chlorite, epidote, kaolin, leucoxene, and hematite is a very pronounced characteristic. Later

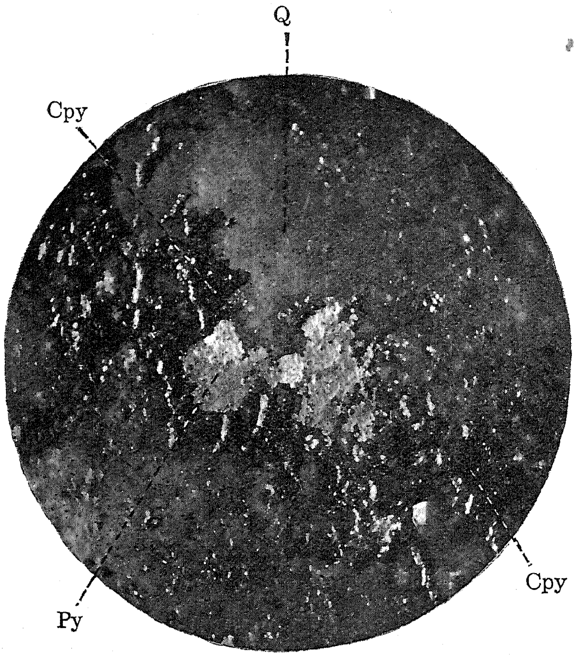


Fig. 6.—Ore. Polished Section Showing Distribution of Pyrite and Chalcopyrite in the Quartz Gangue. Silver Nitrate has been Used to Stain the Chalcopyrite. The Minerals are Disseminated as Replacements in the Quartz, as well as Distributed Along the Veinlets and Channels Between the Quartz Grains. Both Sulphides Appear to have been Deposited at the Same Time. Reflected Light. × 24.

changes have developed some limonite, which is probably derived from pyrite, though occurring chiefly along fractures. Other fractures appear in the slides which are still more recent than these.

The belt of quartz-porphyry, which is the oldest of the rocks, is peculiar on account of the fact that the phenocrysts in it seem to have been subjected to a certain amount of primary fracturing. This would indicate that their original matrix had become somewhat viscous before movement within the mass entirely ceased, although all evidence of flow structure in the matrix or ground-mass has since been obliterated, if ever developed. Specimens of the unaltered rock would undoubtedly

tell this. Nevertheless, its general heterogeneous nature suggests that it was originally a surface flow or large extrusive mass.

The quartz-porphyry within the boundaries of the ore deposit has been thoroughly silicified, and this gray quartzose material is streaked with veinlets and bands of pyrite and chalcopyrite. The original structure of the rock is only slightly preserved and Figs. 4 and 5 will show it to be replaced by aggregates of quartz grains of irregular size and shape.

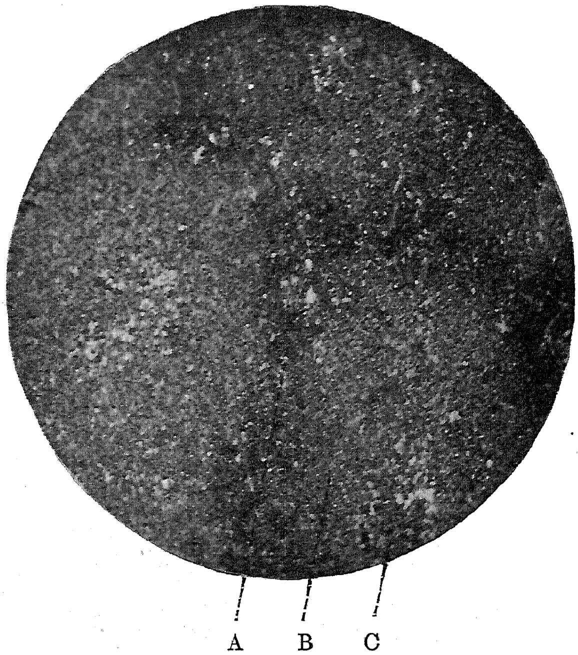


FIG. 7.—SO-CALLED VARIOLITE, A MARGINAL PHASE OF THE QUARTZ-PORPHYRY, MADE UP OF SECONDARY QUARTZ AND ACTINOLITE. THE APPARENT PECULIAR SPHERULITIC STRUCTURE SHOWN IN THE PHOTOGRAPH IS A SECONDARY DEVELOPMENT. A, ACTINOLITE IN FIBROUS PATCHES BORDERING THE FINE QUARTZ AGGREGATES. B, FINE SECONDARY QUARTZ. C, COARSER-GRAINED NUCLEUS OF SECONDARY QUARTZ. CROSSED NICOLS. $\times$ 24.

Veinlets occur which include chlorite and the sulphides. The banded nature of the secondary quartz is also apparent. The interior structure of the quartz grains, their outline and irregular shape, features which are not brought out clearly except under crossed nicols, suggest the replacement of porphyritic material.

The photograph of a polished section of the ore, Fig. 6, shows the distribution of the pyrite and chalcopyrite. The minerals occur partly disseminated as small crystalline aggregates in the quartz mass and partly concentrated along veinlets and channels between the quartz grains. They appear to have been deposited simultaneously, though somewhat later than the period of silicification of the quartz-porphyry.

The chlorite occurs within and around the sulphides as patches, and possibly actinolite along with it as needles, occupying with the sulphides the spaces and channels between the quartz grains. The chlorite is believed to have formed from the ferromagnesians of the original rock, incident with its silicification, and to have had some influence on the precipitation of the sulphides. As a mineral, it is too universal an alteration product to have a more direct significance. The actinolite will be discussed later in connection with the associated rocks.

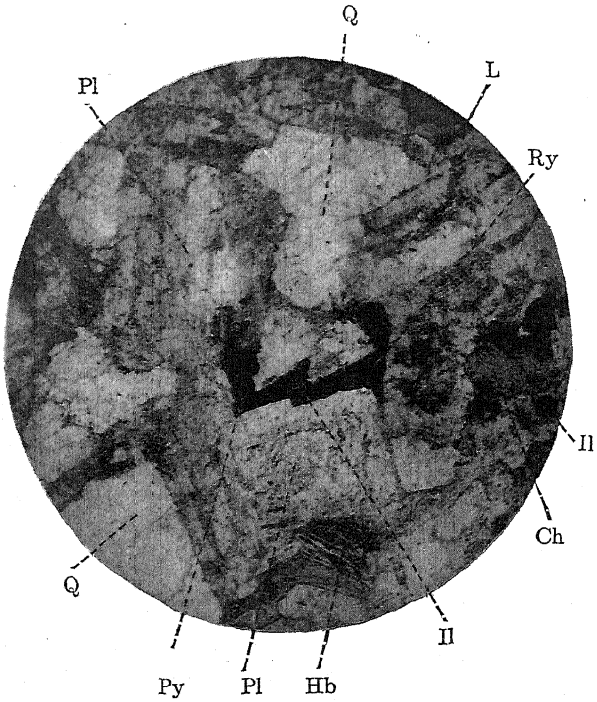


FIG. 8.—QUARTZ-DIORITE, TYPICALLY GRANITOID IN TEXTURE AND MADE UP OF INTERLOCKING GRAINS OF PLAGIOCLASE, QUARTZ, AND HORNBLENDE. ILMENITE AND APATITE ARE ACCESSORY MINERALS. PYRITE OCCURS IN THE ROCK AND MAY POSSIBLY BE PRIMARY, AS IT DOES NOT APPEAR TO CUT THE OTHER CRYSTALS. PLANE LIGHT. × 24.

A peculiar marginal phase of the quartz-porphyry has been developed at the northeast edge of the orebody near the dikes for which the name variolite, see Fig. 7, has been suggested on account of its peculiar spherulitic structure. But variolite is a variety of basaltic glass. The apparent spherulitic structure here is merely a peculiar secondary development of quartz and actinolite.

The Quartz-Diorite

The granite, on account of the predominance of plagioclase feldspar, is best termed a grano-diorite or quartz-diorite. It is a speckled, greenish-

ray and white rock showing abundant hornblende and plagioclase, with uartz, in large interlocking grains. It shows some slight fracturing, and onal growth of plagioclase. The feldspar is basic in composition, probably abradorite, and comparatively free from alteration. Some carboniza-ion is observable in the feldspar, and the hornblende is largely changed o chlorite. Ilmenite and apatite are accessory minerals, some of the patites being 2 to 3 mm. in length. Pyrite occurs in the rock and may ossibly be primary, since it does not appear to cut structures in the other ninerals to any considerable extent (see Figs. 8, 9, and 10).

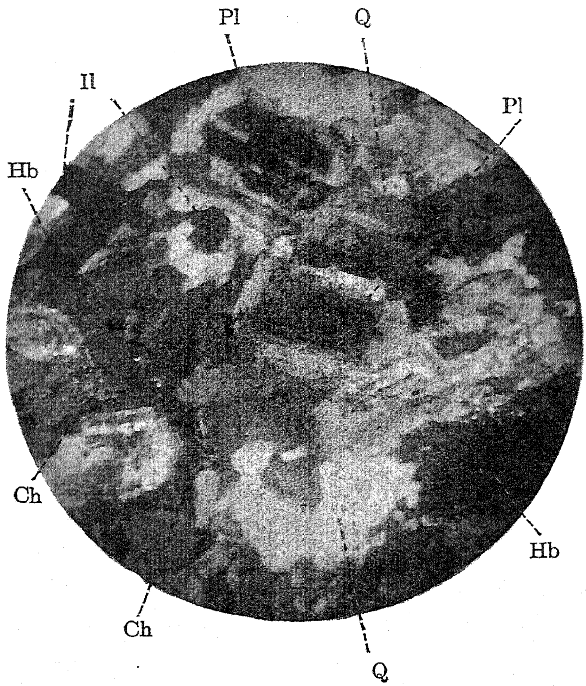


FIG. 9.—SAME ROCK AS FIG. 8, SHOWING ZONAL GROWTH OF PLAGIOCLASE, ALSO THE DEVELOPMENT OF CHLORITE FROM HORNBLENDE. CROSSED NICOLS. × 24.

The "Old Basic"

This body is intrusive into the quartz-porphyry as laccoliths and umerous steep and inclined dikes. It is a dark gray felsitic rock, veathering on its surface to a whitish limonite coating. Though felsitic, t does have a few very small feldspar phenocrysts (see Figs. 11 and 12). The rock is composed of small criss-crossed columnar crystals of plagio-lase with some larger irregular grains of the same mineral. All are urrounded by pale green actinolite and chlorite, derived from altered yroxene, and small grains of magnetite are scattered through the mass.

Quartz is present in small grains but is very largely of secondary origin. Small and sparse crystals of pyrite occur associated with patches of chloritic material and actinolite. The heavy iron content in magnetite attests to the basic character of the rock, the magnetite having also resulted from the alteration of the pyroxene. Evidently the ferromagnesian minerals were unstable under conditions which did not affect the feldspars. The rock is an unusual type, though composed of common minerals, and may be called an andesine basalt.

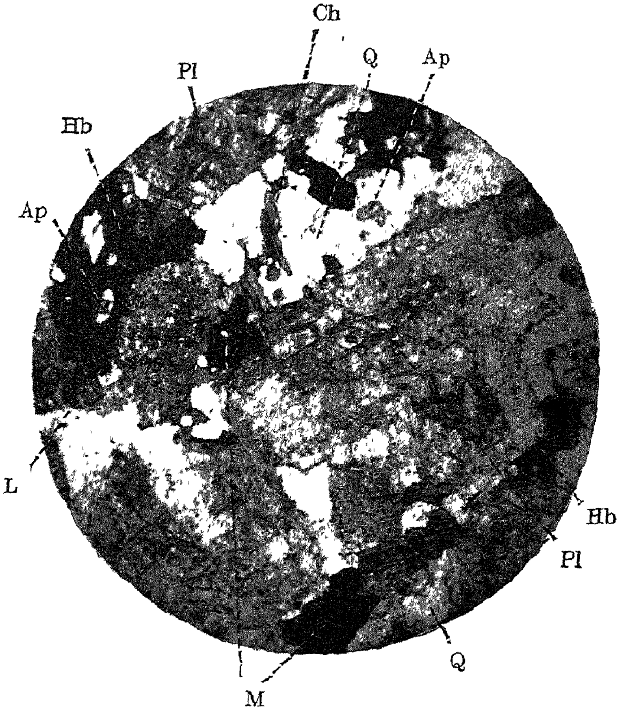


FIG. 10.—THIS PHOTOMICROGRAPH IS SIMILAR TO FIG. 8. IT SHOWS ABUNDANT PLAGIOCLASE, QUARTZ, AND HORNBLENDE. MAGNETITE, POSSIBLY ILMENITE, IS A PROMINENT ACCESSORY MINERAL, AND APATITE CRYSTALS OF FAIRLY GOOD SIZE ARE SCATTERED THROUGH THE ROCK. ALTERATION HAS AFFECTED MOST OF THE FELDSPARS TO SOME EXTENT, ATTACKING THEM ALONG STRUCTURE AND FRACTURE LINES. PLANE LIGHT. $\times$ 24.

Fig. 13 illustrates one of the thin sections under high power, and shows very clearly the small fibrous crystals of actinolite associated with the magnetite. As above stated, the actinolite, and probably most of the magnetite, is regarded as a metamorphic derivation from the original pyroxene in the rock, pseudomorphs of which are common through it. Pyrite also occurs in places penetrated by crystals of the actinolite, tending to the belief that this mineral, as well as the pyrite, is made up partly

of introduced material. However, the pyrite may have formed around the actinolite at a later stage without replacing it.

Stope Dike

This rock, Fig. 14, belongs to the older dike series. It is dense, black and felsitic, very much resembling the "Old Basic," if indeed it is not the same rock. It includes an occasional fine patch of minute pyrite crystals, and weathers with a reddish iron stain. The structure of the

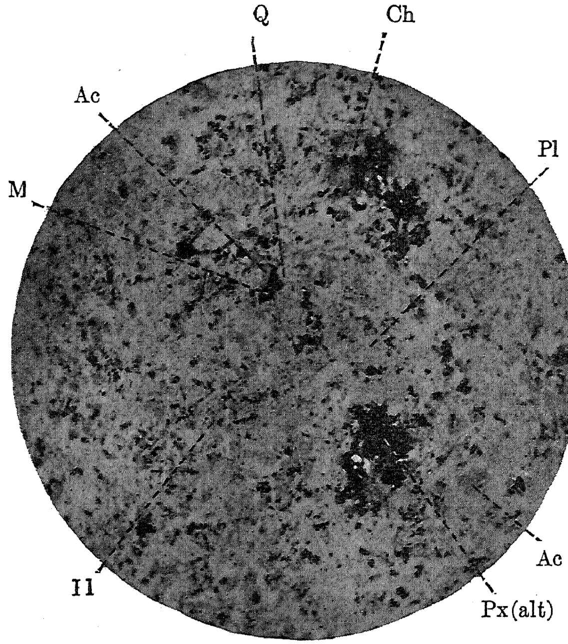


FIG. 11.—ANDESINE BASALT, SHOWING PYROXENE ALTERED TO SCATTERED AGGREGATES OF MAGNETITE WITH ACTINOLITE AND CHLORITE. FELDSPAR IS PROMINENT AND APPEARS TO HAVE BEEN THE FIRST MINERAL TO CRYSTALLIZE. ITS VARIETY IS ANDESINE. QUARTZ, LARGELY SECONDARY, IS PRESENT AS SMALL GRAINS THROUGHOUT THE MASS. THOUGH COMPOSED OF COMMON MINERALS, THIS IS AN UNUSUAL TYPE OF ROCK. PLANE LIGHT. $\times$ 24.

rock may be described as a network of small columnar crystals of plagioclase and fibers of actinolite, filled in with irregular grains of quartz and magnetite. A few fine fractures appear which are healed with chlorite. Pyroxene was an original mineral, which, together with the feldspar, has been largely altered to quartz and actinolite.

Dolerite Dikes

The material of these dikes varies slightly in composition and greatly in texture, some being diabasic and others basaltic. All the material

is dark, massive and more or less felsitic. Fig. 15 shows the diabasic type. It is made up of rounded irregular grains of augite in a plexus of lath-shaped crystals of feldspar. The feldspar is plagioclase. The augite grains are small and irregular in outline, and appear to fill the spaces between the crossed crystals of plagioclase. Alteration has changed a considerable portion of it to chlorite and actinolite, part of which has replaced occasional lines and twinning bands in the plagioclase. Alteration has also changed some of the plagioclase to quartz. This

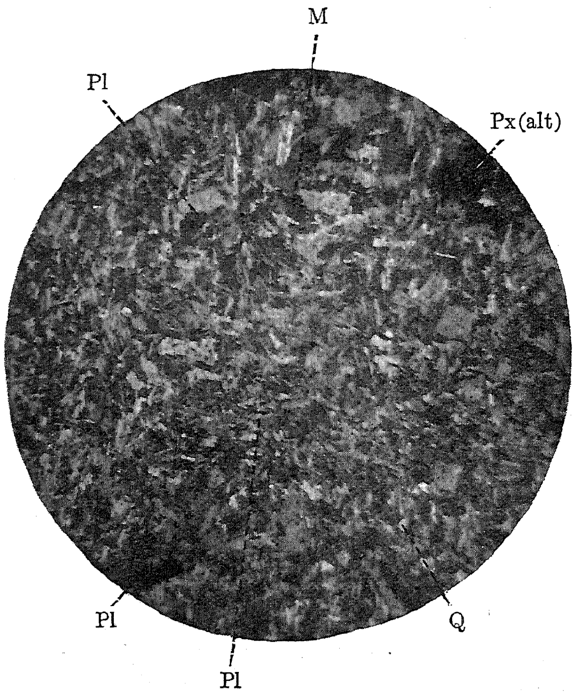


FIG. 12.—SAME ROCK AS IN FIG. 11. THE TEXTURE IS COARSELY FELSITIC TO PORPHYRITIC, AND THE CRISS-CROSSED ARRANGEMENT OF THE COLUMNAR PLAGIOCLASE CYSTALS IS APPARENT. THERE ARE A FEW PLAGIOCLASE PHENOCRYSTS VISIBLE, AND MAGNETITE IS DISTRIBUTED THROUGHOUT. CROSSED NICHOLS. × 24.

quartz seems to have been derived from both pyroxene and feldspar, being penetrated by fibrous actinolite crystals of simultaneous growth. However, it can not be said that all the quartz is secondary, for in the case of many interstitial grains, the traces of a replaced mineral are too slight to make it possible to say it existed.

The term Dolerite, as used on the map, while indicating rock of this general character, does not sufficiently emphasize the important structural feature shown in the arrangement of the feldspars in this particular specimen, and it would, therefore, be more properly called Diabase. The idiomorphism of the feldspars with respect to the pyroxene, as shown by

Fig. 15, is also a distinguishing characteristic of diabase, along with their columnar structure and crisscross arrangement.

Similarly, for textural reasons, the dolerite shown in Fig. 16 is more properly called basalt. This material is a more felsitic variety of the rock composing this group of dikes. It seems to possess the same general alteration effects except that the quartz, if present, is not visible, which might argue that the quartz of the other basic rocks is entirely secondary. In the ground-mass are many fine rods of plagioclase, but as a whole it

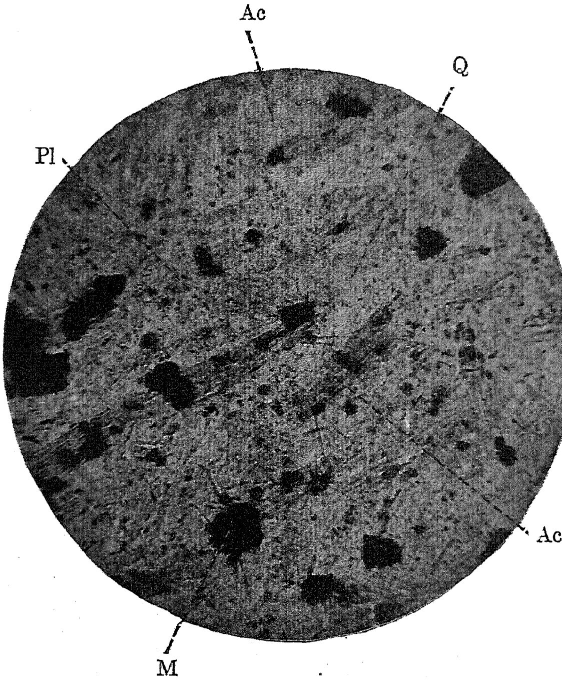


FIG. 13.—SAME ROCK AS FIGS. 11 AND 12. THE HIGHER POWER REVEALS SMALL FIBROUS CRYSTALS OF ACTINOLITE ASSOCIATED WITH THE MAGNETITE. THE QUARTZ AND PLAGIOCLASE ARE RATHER INDISTINCT. THE ACTINOLITE, AND PROBABLY MOST OF THE MAGNETITE, IS BELIEVED TO BE DERIVED FROM THE BREAKING DOWN OF ORIGINAL PYROXENE UNDER THE ACTION OF SILICEOUS MAGMATIC WATERS. PLANE LIGHT. × 257.

is finely felsitic and the rock a typical basalt. It contains some pyrite, which is generally taken to be an introduced substance.

The rock of the East Dike is a dark massive diabase, shown in Fig. 17. Quartz occurs in much the same relation to the crisscrossed columnar plagioclase as the pyroxene, and because this quartz may be partly primary the rock might well be called a quartz-diabase, which is a rather rare rock. In this original, the augite-pyroxene and plagioclase have been largely changed to chlorite, actinolite, and secondary quartz. The quartz and the unaltered plagioclase are penetrated to some extent by

fibers of actinolite. The scattered magnetite is probably also secondary, and pyrite is present.

The variation in texture of these dike rocks is mainly due to different rates of cooling, occasioned either by the influence of wall rock, which may have been hot or cold, or by the thickness of the dikes themselves. Their chemical composition must also have exerted a strong influence on the order of crystallization of the minerals, producing the tendency to idiomorphic development of the feldspar.

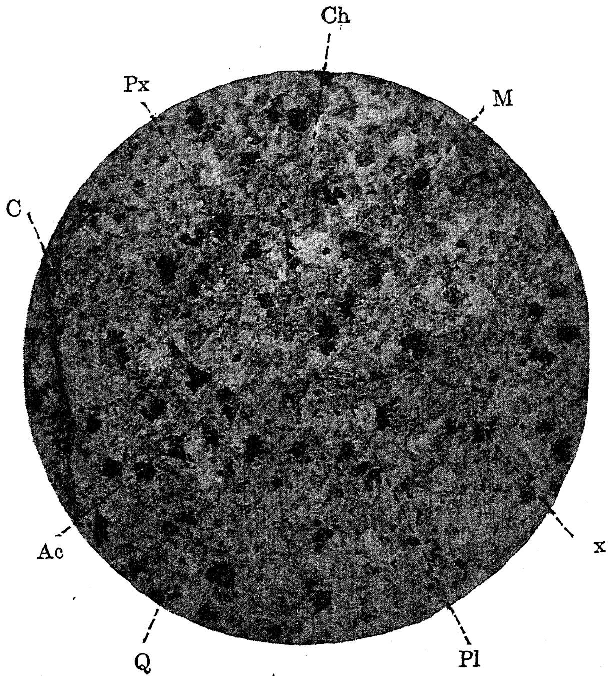


Fig. 14.—Quartz-Basalt, a Felsitic Complex of Plagioclase and Pyroxene Which has been Partly Altered to Quartz, Actinolite, and Magnetite, Leaving Only a Small Part of the Plagioclase Intact. Part of the Quartz, However, Appears to be Original. Abundance of Magnetite, Part of Which is Also Primary. Small Carbonate Vein. Plane Light. × 24.

Summary

It will be seen from the descriptions above that, aside from the quartz-porphyry, the principal rocks associated directly with the ore are basaltic in character. They contain quartz, part of which I judge to be primary and part secondary. Magnetite is widely distributed and appears to have resulted in large part from the breaking down of the original ferromagnesian minerals. The rocks have undergone great change, but do not appear to have been much affected by superficial weathering.

The strong development of actinolite in all the specimens indicates

either the alteration of pyroxene and plagioclase by solutions, or their change to a more stable mineral under conditions of regional metamorphism. The first is more probable, since there is not the slightest evidence of metamorphism in the surrounding grano-diorite. The pyrite in the basic rocks appears to be later than the actinolite, but as a whole is probably associated with the same period of alteration by hot solutions, the phenomena of expiring vulcanism. It is not known whether this pyrite contains gold or copper. In respect to the actinolite, it is not

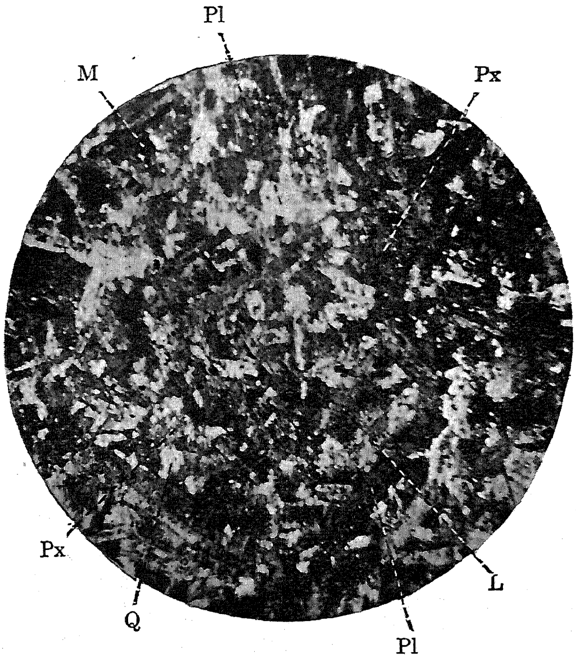


Fig. 15.—A Diabase with the Characteristic Texture. The Augite Grains are Small and Irregular in Outline. Much of the Pyroxene has been Changed to Chlorite and some Actinolite. Twinning of the Columnar Plagioclase Crystals can be Seen. The Quartz is Probably Derived from both the Pyroxene and Feldspar, Being Penetrated by Fibrous Crystals of Actinolite of Simultaneous Growth. The Name Dolerite is as Good a One as Any, but the Usage Varies Among Petrographers. Crossed Nicols. × 24.

possible to say that the alteration is due to direct igneous influence; but because actinolite is generally regarded as a high-temperature mineral there is the possibility of regarding it in much the same light as tourmaline, on the presence of which in the volcanic copper deposit at Braden, Chile, the primary character of the deposit was regarded as certain.

The quartz in general seems also to be a result of metamorphic action on the plagioclase and pyroxene, penetrated as it is by the contemporaneous crystals of actinolite. Part of the quartz in the dikes, however, must be primary, as the sharp outline of some of the interstitial grains

involve difficulty in conceiving the replaced original if these particular quartz grains were secondary. Hence the basic rocks might well be called quartz-basalts and quartz-diabases, which rocks are rather rare occurrences.

The grano-diorite or quartz-diorite intrusion shows some evidence of superficial weathering, but otherwise no alteration, and is considered a fairly fresh igneous rock. Pyrite is present in it, and the more altered character of the rock near the quartz-porphyries has led to the belief that its intrusion had something to do with the formation of the ores.

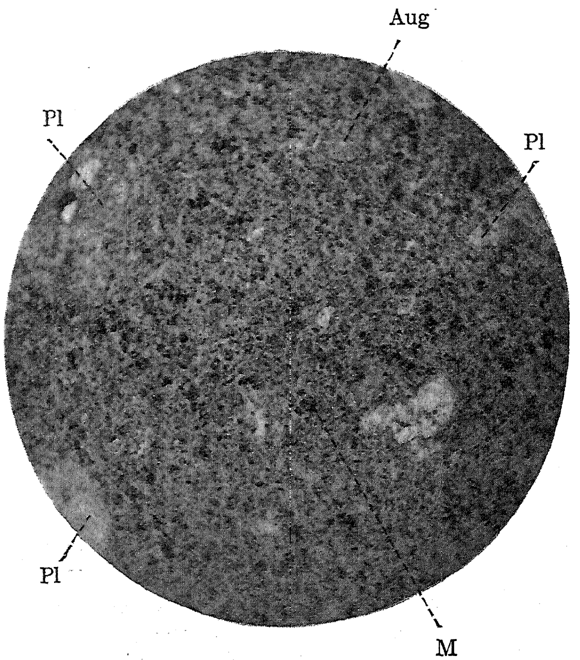


FIG. 16.—BASALT, A MORE FINELY FELSITIC VARIETY OF THE FOREGOING ROCKS, BUT CONTAINING SMALL SCATTERED PHENOCRYSTS OF PLAGIOCLASE. IN THE GROUND-MASS THERE ARE MANY LITTLE FINE RODS OF PLAGIOCLASE. QUARTZ, IF PRESENT, IS NOT VISIBLE. PLANE LIGHT. ×24.

The quartz-porphyries are thought to be such, rather than volcanic tuffs, because they contain no material foreign to the composition of a crystalline igneous rock, except the secondary quartz. By this quartz, however, these rocks are almost entirely replaced, the ground-mass completely so, and on this account their appearance under the microscope does somewhat resemble that of siliceous sinter. From the specimens examined and the mapping of this mass as quartz-porphyry, jasper, etc., its heterogeneous character, as of acid extrusive rock in general, is apparent. One of the specimens is a rhyolitic porphyry and another dacitic, and the structure of both, as already described, also suggests that the mass is an

ancient surface flow of large extent. The later intrusions of quartz-diorite and the basic rocks have further altered its local character.

In regard to the sequence of the different igneous masses, it seems most reasonable from this study to put the "Old Basic" later than the grano-diorite and of the same age as the Stope Dike. No very strong structural evidence to the contrary is shown on the map, as the two apparent intrusions of grano-diorite into the "Old Basic" on the western and southern borders of the mineralized area may just as well be interpreted as

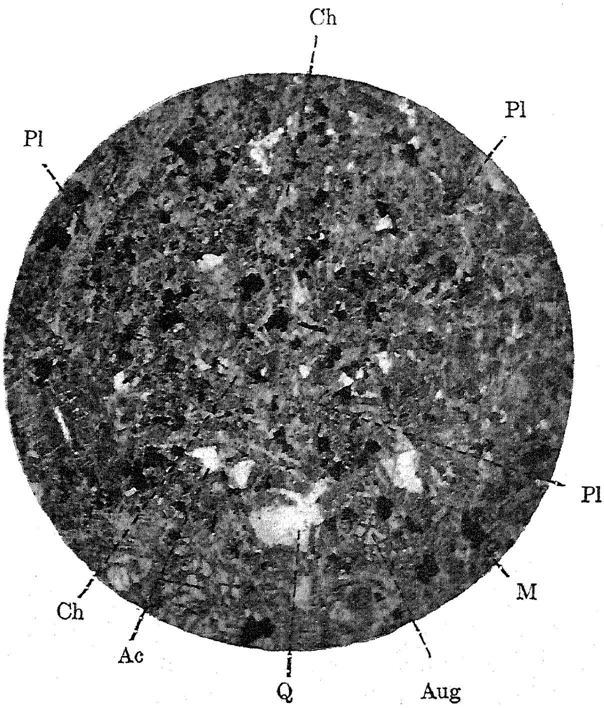


FIG. 17.—ORIGINAL QUARTZ-DIABASE IN WHICH THE AUGITE AND PLAGIOCLASE HAVE BEEN LARGELY CHANGED TO CHLORITE, ACTINOLITE, AND QUARTZ. THE DIABASIC TEXTURE IS EVIDENT, PYROXENE AND CHLORITE FORMING A FILLING BETWEEN THE CROSSED COLUMNAR CRYSTALS OF PLAGIOCLASE. ACTINOLITE NEEDLES PENETRATE THE QUARTZ WHICH, HOWEVER, IN SUCH A BASIC ROCK, MUST BE LARGELY SECONDARY. PLANE LIGHT. × 24.

intrusions of "Old Basic" around spurs of the grano-diorite, and the petrographic evidence we have in addition is very strongly in favor of placing the "Old Basic" in the same age relation to the grano-diorite as the two systems of basic dikes which cut it. All the basic-rock types show too marked a mineralogical and textural similarity, even in the character of their alteration, to be divided into two periods of intrusion by another rock of opposite character, the grano-diorite. However, such occurrences are not unknown in the history of rock magmas.

According to this interpretation, the relative ages of the different masses are as follows:

Formation	Rock Name
9. Sedimentaries—Mesozoic.	
8. Faults.	
7. "East" Dike (Dolerite).	Quartz-diabase.
6. "North" and "South" Dikes (Diabase).	Quartz-basalt.
5. Andesite Dikes.	
4. "Old Basic," "Stope," "East and West" and "Cross" Dikes.	Andesine- and quartz-basalt.
3. "Flat" Dike(?).	
2. Granite.	Quartz-diorite.
1. Porphyry.	Quartz-porphyry.

The ore is of granular secondary quartz containing pyrite and chalcopyrite with chlorite and actinolite, a highly silicified phase of the quartz-porphyry. No evidence of a brecciated or crushed zone appears in any of the slides studied. The enrichment has been derived from magmatic solutions, and there is no reason to believe that the pyrite and chalcopyrite were deposited at different times. The more abundant occurrence of the sulphides along spaces or channels between the grains of quartz, and in bands or streaks through the gangue, seems to indicate that they were deposited shortly after the quartz replacement, rather than at the same time. The associated chlorite and actinolite seem to have been derived mainly from the ferromagnesians of the original quartz-porphyry during the period of replacement by quartz.

The chlorite seen in the ore is, as stated before, believed to have no important significance, but referring to Figs. 2 and 3, the strong development of actinolite in the surrounding quartz-porphyry is believed to be closely associated with the sulphide mineralization. If such is the case, the presence of this mineral in the basic dikes might imply that they had to do with the enriching solutions rather than the earlier grano-diorite intrusion, as actinolite is a metamorphic mineral formed either as a result of hot water action or of direct igneous metamorphism. It is not uncommon to find it at the contact of basic with quartzose rocks, as, for instance, near the peridotite dikes in the Franciscan sandstone of California, which formation itself is highly metamorphosed.

It might be noted here also that many gold deposits are associated with basic rocks, and the further structural fact can not escape attention that the ore deposit here occurs near the intersection of two systems of basic dikes. Hence, to say the mineralization was contemporaneous with these dikes seems a closer correlation of the data than to associate it with the earlier grano-diorite, for there are numberless other points in the quartz-porphyry near the grano-diorite contact where enrichment might have occurred and did not do so.

Discussion

L. C. Graton, Cambridge, Mass.—May I ask, in connection with one point in the summary, to what extent the author regards actinolite equivalent to tourmaline as an index of high-temperature and presumably magmatic conditions of deposition of the accompanying sulphides?

W. E. Gaby.—I merely advanced the idea that the mineral actinolite might have some such significance. In the copper deposit at Braden in South America, the ore occurs in an andesite breccia which is cemented by a matrix of sulphides, quartz, and tourmaline. Tourmaline is a high-temperature mineral, and when we find it associated with ores, these are regarded as having come from a volcanic or deep-seated source, hence a primary hypogenetic deposit. Thus we can count on greater continuation of the ore with depth than in some other kinds of deposit, and I thought that the actinolite, being also a high-temperature mineral, might have some such bearing in the case of Mount Morgan. I did not advance this idea as a final conclusion from a study of such phenomena, but thought that it might bear some discussion in view of the fact of the peculiar occurrence of this mineral in a notable orebody. It also occurs with the copper ores at Ducktown, Tenn., and in both cases may only be due to general regional metamorphism.

Geology of the Warren Mining District

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CONTENTS

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I. INTRODUCTION

THE main object of this article is to present the results of observations by the Copper Queen Consolidated Mining Co.'s geological department as an addition to the already published reports on the Warren district. Since these observations relate mostly to the ore deposits, little will be found here on the general geology that has not been covered by F. L. Ransome's admirable work on the Bisbee Quadrangle.[1] A summary of this very important part of the study of the camp will be given, however, mainly to bring out the new facts discovered during the advance in underground development and the latest detailed study of structures and petrography of the mining area.

II. PHYSIOGRAPHY

The Mule Mountains, in which the Bisbee ore deposits occur, constitute a chain with a northwest-southeast axis rising abruptly on the sides from the Sulphur Springs Valley on the northeast, as shown in Plate 1, and the San Pedro Valley on the southwest. The range starts at about the International Boundary near Christianson's ranch, with low hills covering a width of 1 or 2 miles, and extending for about 3 miles to the northwest. Here the Gold Hill overthrust fault has caused an abrupt rise in elevation and from this point on the range widens out and the hills become very much more rugged. The highest point is about 6 miles farther. The total length of the range from the International Boundary is about 23 miles and its maximum width, about opposite the town of Bisbee, is 10 miles. The range finally ends at Government Draw, which is a pass about 2 miles wide, connecting the San Pedro and Sulphur Springs Valleys. Beyond this the Tombstone hills commence.

The range is divided into two parts by Tombstone Canyon, a deep canyon running through the southwest side of the range. It is along this canyon that the Borderland Route road takes its course, and in which the town of Bisbee is situated.

Geologically, also, as seen in Plate 1, the Mule Mountains are roughly divided by Tombstone Canyon into two parts. To the southwest is the pre-Cretaceous tract and to the northeast the Cretaceous. The difference in the two tracts is very marked physiographically. The pre-Cretaceous is very much cut up and is formed of rocks of diverse compositions, consisting of schists, granites, shales, and limestones. These rocks, when subjected to erosion, form a rugged topography with deep canyons and steep cliffs. The Cretaceous tract, on the contrary, is uniform in composition, constituted, for the most part, by soft sandstones, conglomerates,

[1] Geology and Ore Deposits of the Bisbee Quadrangle, *Professional Paper of the U S. Geological Survey,* No. 21 (1904).

and shales, which are but little faulted. The topography carved from them is characterized by gently sloping hills and draws with almost no cliffs or deep valleys in evidence.

III. INTRODUCTORY GEOLOGY

The oldest rocks of the range are pre-Cambrian schists and an intruded granite, which are separated from the overlying Paleozoic beds by a profound unconformity.

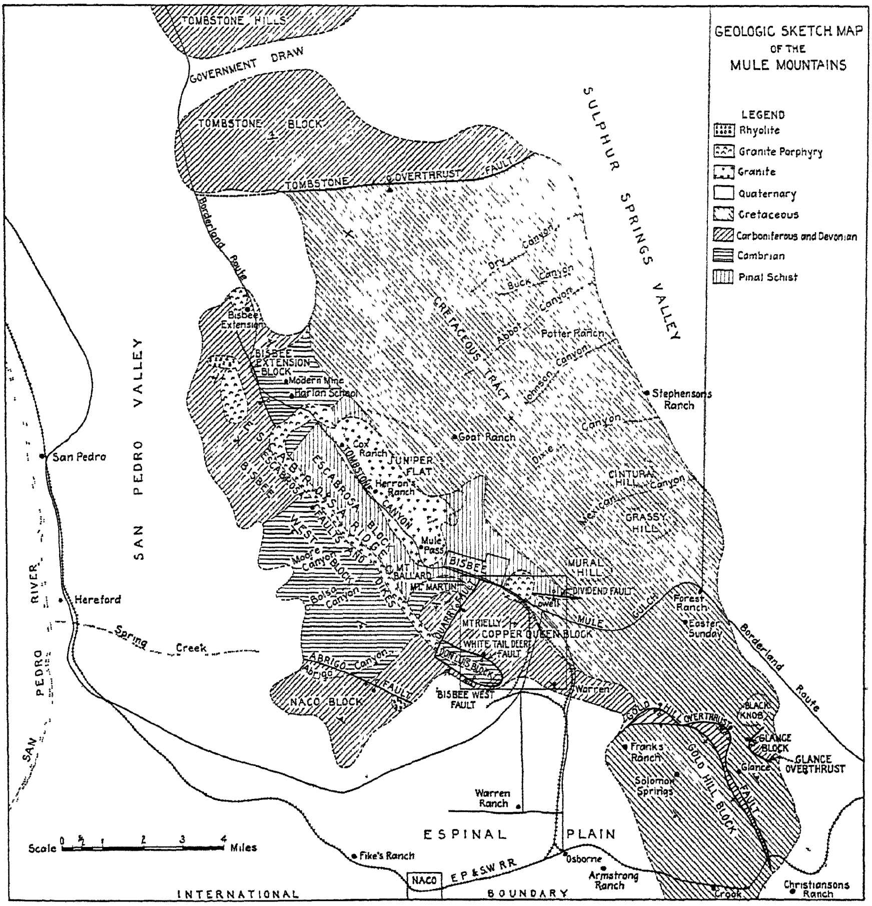

PLATE 1.—GEOLOGICAL MAP OF THE MULE MOUNTAINS.

The Paleozoic beds represent an era of apparently uninterrupted deposition of sediments, starting with 440 ft. of quartzites having a basal conglomerate, followed by Cambrian limestones, Ordovician-Silurian quartzite, Devonian and Carboniferous limestones. The total thickness of the Paleozoic beds is approximately 5,000 ft.

At the end of the Carboniferous era a violent uplift took place accompanied by extensive faulting and intrusion of granite porphyry. During this uplift, and intimately associated with the intrusion, mineralizing solutions arose from which originated the orebodies of the camp.

Following this came a long period of erosion, which ended with a rapid subsidence, during which time 4,500 to 5,000 ft. of Cretaceous sandstones, shales, and limestones were deposited.

At the end of Cretaceous time a gentle uplift took place, lifting the range again above sea level. This was accompanied by some intrusion of rhyolite and monzonite.

During Tertiary and Quaternary ages, this land area was subjected to erosion, resulting in the present topography, the detritus filling in both the valley of the San Pedro and of the White River, one on each side of the range.

IV. ROCKS OF THE DISTRICT

A. Sedimentary Rocks

Reference to Plate 2, showing a generalized geologic section, will help make clear the following description.

Pinal Schist

Name.—The oldest rock of the district is the Pinal Schist. It is composed of a uniform series of thinly laminated siliceous mica schists of unknown thickness. From the similarity in texture and stratigraphic relation to the later sediments, it has been correlated by Ransome[2] with the underlying schistose complex of the Pinal Range, which he called the Pinal Schist.

Distribution and General Structure.—The Pinal Schist, being the basal crystalline rock upon which rest all the younger formations of the district, is exposed everywhere that erosion has stripped them off, and is encountered underground on sinking through them.

The best surface exposures are northeast of Mule Gulch and to the southwest of Tombstone Canyon. Smaller exposures are also found about 1 mile to the northwest of the town of Don Luis. Underground it is exposed where the Dividend Fault is cut, both in Copper Queen and in Denn ground. It also is found as dragged-in fragments in the contact-breccia mass around Sacramento Hill.

Lithology.—The color varies from light to dark gray, with tinges of green on fresh surfaces, and rusty in weathered specimens. The cleavage is rather imperfect, having a shiny satin-like surface.

The microscope shows the rock to be composed mainly of quartz and sericite. Sometimes the sericite is replaced in part by penninite, chlorite,

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[2] *Loc. cit.*, p. 24.

or serpentine. Folded into the schist, and forming an integral part of it, are occasional masses of diabase.

Origin and Age.—The age of the Pinal Schist is assumed to be the same as for the Pinal Schist of the Globe district which Ransome[3] has shown to be derived from arenaceous sediments, probably of Algonkian age. Both at Globe and at Bisbee a profound unconformity exists between it and the overlying Paleozoic beds.

COLUMNAR SECTION

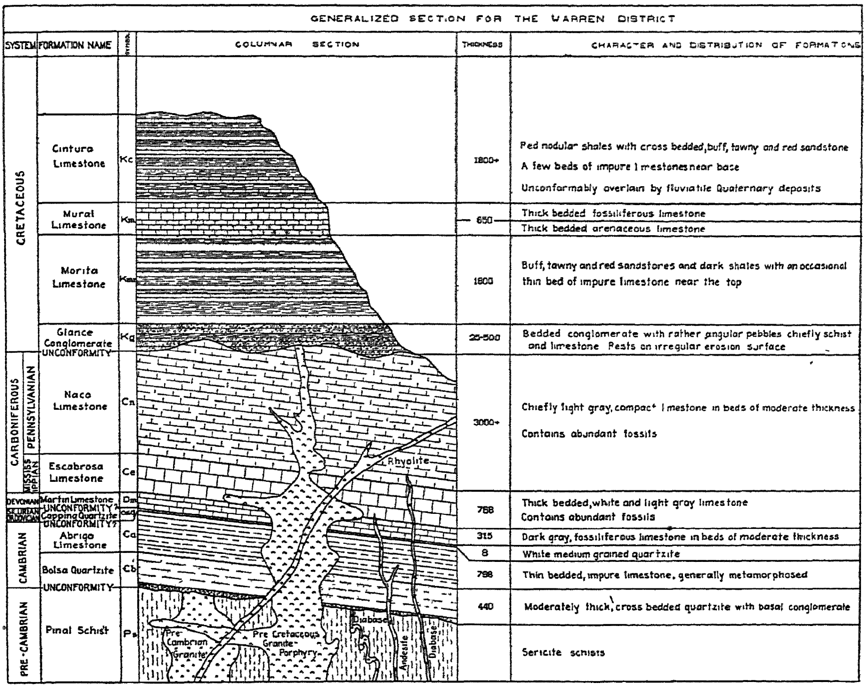


PLATE 2.—GENERALIZED GEOLOGIC SECTION SHOWING ROCK STRATA OF THE WARREN DISTRICT.

Bolsa Quartzite

Name.—The name of this formation, like those for all other formations in the district, was given by Ransome[4] and has been generally adopted.

Distribution and General Stratigraphy.—This Bolsa Quartzite lies on an evenly eroded plain of Pinal Schist. The main exposures of this formation are southwest of Escabrosa Ridge, especially in Bolsa, Abrigo, and Quarry Canyons. In the last-named canyon a complete unfaulted

[3] *U. S. Geological Survey*, Globe Folio, No. 111, p. 2.

[4] *Loc. cit.*, p. 28.

section occurs. Underground, the formation has been cut by diamon drill holes from the bottom of the Irish Mag shaft, and also by the worl ings from the Wade Hampton shaft.

Lithology.—At the base of the formation is a well-marked basal co glomerate, much iron-stained, containing pebbles consisting mostly white, well-rounded quartz of up to a 4-in. diameter, and averaging abou 1 in., the cementing material consisting of subangular sand. In tl northwestern part of Tombstone Canyon some pebbles of porphyrit rock are found, together with some rounded crystals of feldspar. The feldspar crystals consist of pink microcline and microperthite, such are found in the Juniper Flat granite less than a mile away. The feldspars, notwithstanding their being subject to such long exposur are still relatively fresh.

The next 100 ft. of the Bolsa shows alternating beds of fine conglome ates and quartzites, with some arkose beds. Then follows a finer whiti quartzite with frequent cross-bedding, succeeded by some massive, den beds of a maroon-colored quartzite. The total thickness of the Bol Quartzite as measured in Quarry Canyon is 440 ft.

Age.—No fossils have been found in the Bolsa except sparse wor tracks near the top, but as the next formation lies conformably on top it and shows middle Cambrian fauna, the Bolsa has been assumed lower or middle Cambrian.

Abrigo Limestone

Distribution and General Stratigraphy.—The Abrigo formation follo the Bolsa Quartzite conformably, and is the first fossil-bearing formati in the district.

The best exposure of Abrigo limestone is found on the northern slo of Mount Martin, where a complete unfaulted section is obtainab Large areas of Abrigo are also found on the southwestern slope of Esc brosa ridge in Abrigo, Moon, and Bolsa Canyons, but the formation is he badly faulted. Small areas are also found in Quarry Canyon, nor of Don Luis, and on the southern slope of the range from Mount Reil to Gold Hill. As the beds are, for the most part, soft shales and sha limestones, good exposures are rare, the formation being usually cover by talus.

Underground, the Abrigo formation is cut by the lower levels of the mines from the Higgins to the Briggs, but in most of them only t top 100 ft. of rock has been cut. At the Spray, Oliver, and Irish M shafts, however, the formation has been cut by their lower workings a depth of 400 ft. below its top, and diamond-drill holes from the Iri Mag shaft have penetrated into the Bolsa. At the White Tail Deer a Wade Hampton shafts the lower beds have also been cut. No comple section, however, has been exposed underground. Measured at Quar Canyon the thickness of the Abrigo was found to be 798 ft.

Geological section of Abrigo at Quarry Canyon:

Thin shales, greenish-gray to pink-brown, highly epidotized	0–72
Gray broken calcareous limestone, epidotized	72–83
Shaly calcareous sandstones	83–112
Medium-grained greenish-gray limestone with epidotized beds	112–132
Pure fine-grained pinkish-gray limestone	133–134
Arenaceous pinkish-gray shales	134–136
Purer fine-grained dark-gray limestone	136–139
Light-green shales	139–141
Fine-grained limestone, thin wavy bands of epidote	141–146
Green shales	146–151
Succession of impure lime and shales	151–183
Fine-grained gray limestone, beds of epidote ½ to 1 in. thick	183–229
Purer light-gray limestone	229–245
Coarse sandy limestone with epidotized beds	232–245
Thin greenish shales	245–253
Coarse limestone, very fossiliferous (worm tracks)	253–261
Arenaceous green limestone	261–278
Coarse limestone, thin epidotized bands, some purer, 2-ft. beds forming steps on surface	278–412
Impure limestone green to brown, thin wavy bands of epidote very characteristic on surface by the weathering out of the epidote	413–497
Shaly arenaceous limestones	497–521
Coarse gray arenaceous limestone	521–629
Coarse light-gray crystalline sandy limestone	629–684
Coarse cherty-banded crystalline white bed-forming cliff	684–700
Sandy even-grained limestone and quartzite	700–798

The Abrigo formation consists of a series of shales and argillaceous and sandy limestones, with bands of epidote. These epidote bands weather out on exposure to the atmosphere, giving the formation a characteristic cherty banded appearance. In the top beds there are a few bands of true chert. Underground, this epidote appears as greenish bands giving a wavy banding which readily distinguishes it from all other formations, when the beds are fresh. About 100 ft. from the top there exists a 16-ft. bed of pure white limestone which forms a rather prominent cliff when encountered on the surface.

Capping Quartzite

Lying on top of the Abrigo, and with no apparent angular unconformity, is a persistent bed of pure quartzite varying in color from white to dark red, and varying from 6 in. to 12 ft. in thickness, and locally called the Capping Quartzite.

Distribution.—This quartzite being more resistant than the underlying and overlying beds, is exposed wherever the Abrigo outcrops, as a small cliff or shelf. One well-marked exposure is found to the east of Black Gap, where it runs up the side of the hill apparently as a siliceous dike. Other good exposures are found on Mt. Martin, and southwest of Escabrosa Ridge.

Underground, this bed is important as a marker. Due to its character, it is resistant to metamorphism, and may be recognized easily even when the rest of the sediments have been completely changed. Its thinness, however, tends to make it elusive, as a small fault frequently will throw it out of an otherwise perfect section.

Age.—As one travels north through the State, this formation becomes thicker, and in the Globe district it attains a thickness of 500 to 700 ft., while that of the underlying limestone has decreased to 200 ft. Here there is a well-marked evidence of an unconformity between the limestone, which is capped by a flow of basalt of varying thickness, and the overlying Troy Quartzite, which is conglomeratic at its base. While no fossil evidence is available, its relation to the overlying and underlying formations seems to indicate that it is probably of Silurian age, and that the Ordovician is represented by a period of erosion, when the area studied was very little above sea level, since there is no apparent angular unconformity below or above this formation.

Martin Limestone

Lying conformably above the Capping Quartzite is 300 ft. of limestone known as the Martin limestone.

Distribution.—The exposures of the Martin limestone are widespread, especially along Escabrosa Ridge, north of Moore Canyon, in Abrigo Canyon, Escacado Canyon, and on the southern slopes of the hills east of Mt. Reilly. Owing to the fact that the overlying Escabrosa limestone is a prominent cliff-forming bed, the exposures are apt to be talus-covered. The best complete sections are obtained on the northern slope of Mt. Martin, and to the east of Black Gap. Underground, it is the formation most frequently cut of any in the ore zone. For this reason it is usually difficult to recognize from its lithological characteristics. The formation is so rich in fossils, however, that even when much altered, a diligent search will yield some trace of them. The formation is encountered underground in every mine in the district except the Wade Hampton, which is entirely in Cambrian, and the Denn which did not sink deep enough to strike it. The outcrop of this formation was first found in the Queen opencut near its top, and it has been encountered at progressively lower levels from here down to the 1,600-ft. level of the Lowell, and the 1,500-ft. level of the Briggs.

Lithology.—The Martin limestone is the most constant formation, both in respect to thickness, and character, of any in the district. Plate 3 shows a comparison of sections obtained over widely scattered parts of the district. As is seen, the formation as a whole is made up of fairly dense dark-gray limestone with occasional shaly and sandy members, but all with enough lime present to be called limestone. It is all more or less fossiliferous, and some beds are extremely so. The characteristic fossils are three brachiopods, one a spirifer, one an atrypa, and

one a schisophoria, which all attain a length of about ⅓ to ½ in.; they tend to weather out on both surface and underground exposures. When the rock is altered, these fossils tend to resist metamorphism to a

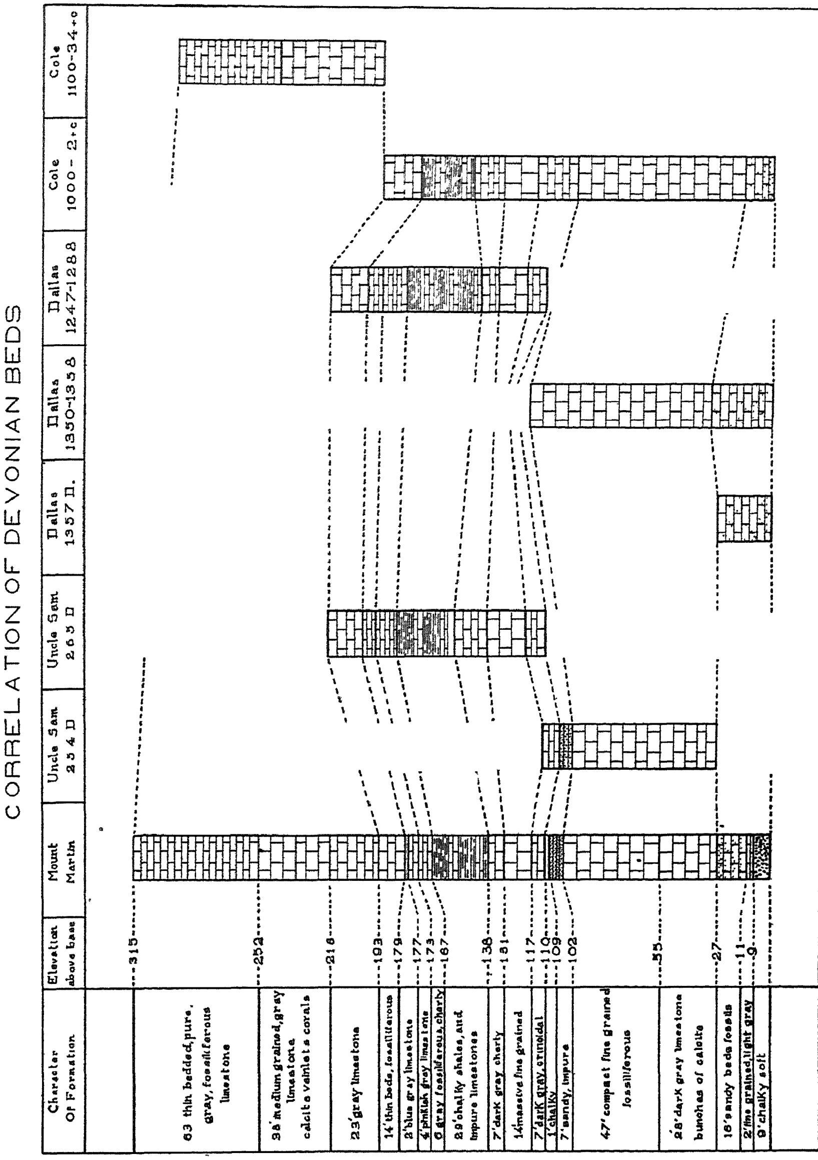


PLATE 3.—COMPARISON OF SECTIONS OF MARTIN LIMESTONE IN VARIOUS PARTS OF THE WARREN DISTRICT.

remarkable degree, and it is not at all rare to find almost perfect fossils left in a soft mass of completely altered limestone. These fossils, however, are frequently found completely pyritized when the surrounding

limestone is untouched. In one case, within a completely oxidized ore-body of the Martin, fossils were found almost perfect in form but completely replaced by azurite. Fortunately, the fossiliferous beds themselves seem to be more easily replaced by ore solutions, so that where metamorphism is most complete, some fossil evidence may usually be found.

The upper beds of the formation have few brachiopods, but are almost made up of corals. On altering, these beds tend to take a pseudo wavy banding, which makes it difficult to distinguish from the Abrigo. The succession of beds in these cases is the only determining factor.

The Martin limestone is not a pure limestone by any means. Almost the whole series is more or less dolomitic, more so than any other formation in the range. It is also much more aluminous than a superficial examination would lead one to suppose. The silica content is also high. Characteristic partial analyses of unaltered specimens of Abrigo and Martin limestones from both surface and underground exposures are given in Table 1. These determinations were made at the Copper Queen assay office.

TABLE 1.—*Analyses of Unaltered Abrigo and Martin Limestones*

	Abrigo					Martin							
	Mt. Martin Section Underground												
	Shales	72–232 ft. above base	232–521 ft above base	521–700 ft above base	Cherty Limestone 1,357 Drift	Black Gap[4]	500 Level Uncle Sam Mine[3]			265[3] Drift	1,300[1] Dallas	1,200[2] Dallas	1,000[5] Cole
SiO_2...	68.6	30.8	35.3	16.1	49.4	10.3	39.4	46.0	21.7	15.5	45.5	35.5	25.2
Al_2O_3..	9.1	4 0	3.0	2.1	4.5	2.0	9.1	10.8	6.5	3.5	12.9	5.0	5.9
MgO...	4.6	3 6	2.3	1.7	2.4	14.2	8.8	4.4	13.8	22.9	3.8	5.4	8.5
CaO...	9.8	33.5	31 5	38.9	17.8	28.5	6.0	10.5	17.8	10.7	13.1	26.7	24.7
Fe. ...	5.1	2 7	2 8	1.5	2.2	1.2	3.7	3.2	2.0	1.8	3.0	2.1	1.8
S	...	....	...	0.2	0.6	0.2	1.5	0.4	0.7	1.3	1.9	0.8	0.8
Mn ...	...	....	....	....	....	....	0.6	0.4	0.4	0.3			

[1] Bottom beds. [2] 100 ft. above Capping Quartzite. [3] Somewhat metamorphosed. [4] Composite of section. [5] 200 to 300 ft. above Capping Quartzite.

As is seen, the two formations are of about the same purity, but the Abrigo contains more silica than the Martin. The Martin, however, is much more dolomitic, and the lower beds are very shaly. The composite from Black Gap shows that on the whole it is a fairly pure dolomite, with only 13 per cent. of silica, alumina, and iron. The impure beds are chiefly at the bottom 100 ft. and the top 50 ft. of the formation.

Age.—The age of the Martin has been well established by good fossil evidence as Devonian. The best exposition on the correlation of these beds is found in the previously mentioned Ransome's report on the Bisbee Quadrangle.

Escabrosa Limestone

Distribution.—The best surface exposures of this formation are on Escabrosa Ridge from Mt. Martin to Mt. Reilly, and on the southern slopes of the hills from Mt. Reilly to Gold Hill. The formation again appears to the northward on Escabrosa Ridge southwest of the Modern mine, and an excellent section is obtained from where the Borderland Route turns out of Tombstone Canyon into the San Pedro Valley. Underground, the formation is cut by all the shafts from the Czar to the Briggs. It is usually in the oxidized portions of the mines, and as a consequence exposures are not very good.

The best exposures found are at the Junction and Sacramento mines. At the Junction, workings cut the formation from the top to the bottom. Here, except for marbleization, the exposures are good, as oxidation has not penetrated nearly as deep as in the other mines of the camp. At the Sacramento, exposures are obtained of the bottom 300 ft., but marbleization has so altered the formation that they are not good.

Lithology.—The Escabrosa and Naco formations represent really one period of deposition during Carboniferous time. The division of the formation as a whole into Escabrosa and Naco is largely an arbitrary matter. Roughly it is divided, from the fossil evidence, into Mississippian and Pennsylvanian, the former locally called Escabrosa, and the latter Naco. This line of subdivision lies at approximately 800 ft. above the top of the Martin formation. In this paper the thickness of the Escabrosa has arbitrarily been taken as 768 ft. irrespective of fossil evidence. Following is a detailed section of Escabrosa taken at Mt. Martin and northeast of the White Tail Deer shaft.

	Feet
Coarse crystalline pink non-fossiliferous limestone	0–6
Pinkish-white to gray medium-grained 6-in. to 12-in. beds	6–60
Massive grayish-white coarse fossiliferous cliff beds	60–140
Thin gray beds medium fine-grained fossiliferous	140–145
Coarse massive grayish fossiliferous cliff bed	145–185
Thin fine-grained fossiliferous gray beds	185–312
Fine-grained grayish-white cherty banded (½ in. to 3 in.)	312–479
Coarse massive white cliff beds fossiliferous	479–556
Fine-grained thin-bedded fossiliferous	556–608
Brown cherty fossiliferous limes	608–616
Grayish fine-grained thin-bedded fossiliferous	616–648
Pinkish beds medium-grained with some chert	648–686
Fine-grained thin-bedded soft grayish	686–758
Massive dark-gray medium-grained	758–768

The Escabrosa limestone rests conformably on the Martin limestone wherever exposed. The 6-ft. pink non-fossiliferous bed forming the bottom member serves in unaltered ground as an excellent marker. The pure coarse crystalline bed following is the most commonly exposed

member of all, forming as it does the steep and rugged cliffs of Escabrosa and Higgins Ridges. At the Mt. Martin and the White Tail Deer sections, it contains very little chert and abundant crinoid stems. Underground exposures appear to contain considerably more chert in this horizon. In fact, the presence or absence of chert beds seems to be very local, and untrustworthy in identifying any particular horizon. Above this first 180-ft. cliff-forming horizon is a soft series of more dense fine-grained beds with some chert, followed by 127 ft. of extremely cherty banded medium-grained limestone, which seems to be very persistent. Above this is a second thinner cliff-forming horizon 77 ft. thick, followed by dense, relatively impure limestone, varying in color from brown to gray with quite abundant chert nodules. There is no definite marker between the Escabrosa and Naco. As previously stated, the Escabrosa is given arbitrarily a thickness of 768 ft.

Naco Limestone

Distribution.—This limestone is generally exposed on the top of the hills from Queen Hill to the east, and extends over the productive area from the Irish Mag shaft continuously east until covered by the later conglomerate Good exposures of Naco are also found at the northernmost range of hills just south of Government Draw, and on Escabrosa Ridge from the Modern mine, to the northwest, where it is finally covered by Quaternary wash. The best and least faulted section of the formation is found in the Naco Hills about 3 miles west of the town of Don Luis. Underground it is cut by all the shafts from the Irish Mag eastwards. Practically no drifts, however, have been driven in this horizon, except possibly at the upper levels of the Junction mine, and the 200-ft. level of the Lowell.

Lithology.—A thickness of 1,500 ft. has been measured in the Naco Hills. Assuming that the formation has been half stripped off we get a probable former thickness of at least 3,000 ft. Compared with the Escabrosa limestone, it is generally thinner bedded, and very much finer grained, and contains some distinctly shaly members, especially in the upper portions. Some coarse-grained beds do exist, however, which without fossil evidence would be impossible to distinguish from the Escabrosa. Abundant crinoids exist as in the Escabrosa, but large brachiopods and cephalopods are much more in evidence. The most common and characteristic Naco fossil is a large-sized member of the productus family.

Cretaceous Deposits

Glance Conglomerate.—Resting on an extremely uneven erosion surface of Paleozoic beds, schist and granite, is deposited the Glance Conglomerate. This formation is exposed over almost the whole of the southern end of the range.

Lithology.—This conglomerate is made up of partially rounded frag-

ments of schist, porphyry, both fresh and sericitized, quartzite, and limestone. These fragments vary in size from ½ in. to 3 ft. diameter and are loosely compacted. Almost no cementing material exists. The conglomerate as a whole is stained a deep brown. The thickness varies from nothing up to 1,000 ft., depending on the old erosion surface. Where this old surface was fairly well worn down, as north of the Dividend Fault and on top of the Juniper Flat granite mass, the formation is a typical basal conglomerate with rounded pebbles cemented with sand.

Age.—The age of the formation is either late Jurassic or early Cretaceous.

Morita, Mural and Cintura Formations.—For a more detailed description of the Cretaceous beds above the Glance Conglomerate, Ransome's report may be referred to. As these beds occur outside the productive area, no additional study was made of them. The Morita consists of a series of buff and tawny sandstones, shales and calcareous sandstones, containing sparse fossil remains, and measuring 1,800 ft. in thickness. This is overlain conformably by 650 ft. of limestone with a 50 to 200-ft. member of pure limestone containing abundant Cretaceous fossils.

The Mural is followed by the Cintura formation, consisting of beds almost identical with those of the Morita, with a thickness of 1,800 ft., plus an unknown amount eroded off. These Cretaceous beds cover about two-thirds of the range, nearly the whole area northeast of Tombstone Canyon and Mule Gulch, and are finally buried beneath the Quaternary wash of Sulphur Springs Valley.

B. Igneous Rocks of the District

The most important igneous rocks of the district, those of granitic composition, have been grouped together in the United States Geological Survey publications on this region, and have all been ascribed to the same magma and general period of eruption, although differences in composition, texture and field relations were noted. Notman[5] has already separated the granite and the porphyritic rocks by placing the first in the pre-Cambrian, and Boutwell has found a basic variety of porphyry underground and on surface, which corresponds probably to the post-Cretaceous dike described by Ransome from the Glance mine,[6] this latter being a rock easily distinguished from the other porphyry.

Careful search for field and petrographic evidence has yielded results which allow the following classification of the igneous rocks of the Warren district.

Pre-Cambrian Granite

On Juniper Flats and along the north side of Tombstone Canyon, a mass of granite is exposed which forms the center of the Mule Moun-

[5] *Transactions of the Institution of Mining and Metallurgy*, vol. 22, pp. 550–562 (1913).
[6] *U. S. Geological Survey, Professional Paper* No. 21, p. 84 (1904).

tains. The rock is a pink or purplish-gray coarse-grained granite whose component minerals are quartz, orthoclase, microcline, a little biotite and plagioclase feldspar. Accessory minerals can be seen only under a microscope in thin section, such as apatite, magnetite, zircon and tourmaline. The orthoclase contains microperthitic intergrowths of albite, and dusty inclusions of iron oxides.

This rock corresponds exactly to that described by Ransome as the Juniper Flats Granite, an analysis of which is given in *Professional Paper* No. 21 *of the United States Geological Survey,* as follows:

	Per Cent.		Per Cent.
SiO_2	75.86	Na_2O	3.60
Al_2O_3	12.17	K_2O	5.04
Fe_2O_3	0.85	H_2O-	0.27
FeO	0.36	H_2O+	0.72
MgO	None	T_1O_2	0.21
CaO	0.62		
			99.70

There are on Juniper Flats some porphyritic alaskite dikes that seem to be directly related to the granite, but aside from these, all other porphyritic rocks here have not been ascribed pre-Cambrian agė.

Field Relations.—The Juniper Flats granite has been found cutting only Pinal Schist, and while it has not been found directly overlain by Paleozoic sediments but covered directly by Cretaceous, pebbles of microcline and of microperthite granite have been found at the base of the Cambrian quartzite. This granite nowhere cuts the Paleozoic sediments, but it is itself extensively intruded by dikes and irregular masses of the later porphyries that do penetrate the sedimentaries. These observations, as well as the similarity of this granite to other pre-Cambrian granites in Arizona, have led to the belief that the intrusion took place before the beginning of the Paleozoic.

In the area economically exploited in the district for copper, the Juniper Flats granite has not been found, and the only ore associated with the rock comes in fissure veins filled with quartz and some fluorite with pockets rich in gold.

Pre-Cretaceous Granite Porphyry

The most important igneous rock in the district is the granite porphyry that is associated with most of the orebodies, and whose largest outcrop is the mass of Sacramento and Copper King Hills in Bisbee.

Distribution.—On surface this rock is found abundantly in the southwest, or older portion, of the Mule Mountains, but the distribution is by no means even. In the schist, granite, and Cambrian formations there are abundant masses of porphyry in the form of dikes and sills, while in the Carboniferous only a few well-marked dikes are found besides the main stock of Sacramento Hill.

In Juniper Flats and on Escabrosa Ridge there is abundant granite

porphyry, some of the dikes cutting granite, schist and Paleozoics in their course from the north to the south side of Tombstone Canyon. On one of the ridges close to the mouth of Tombstone Canyon, about opposite the old Modern mine there is a continuous sill of granite porphyry several hundred feet thick in Carboniferous limestones. This sill is remarkable because it is cut by dikes of rhyolite, and because on an old erosion surface of the granite porphyry and Carboniferous limestones a considerable thickness of rhyolitic lavas has accumulated.

The character of these later dikes and flows is exactly the same as that of the volcanic plug with flow structure which is found in the Naco hills and is mentioned by Ransome in his report on the district. The relations in the locality at the mouth of Tombstone Canyon leave little doubt as to the great difference in age between the granite porphyry and the rhyolite, as the erosion surface on which the rhyolite flowed is probably post-Cretaceous.

In the mining area of the district the Sacramento Hill stock has penetrated between the Paleozoic sediments on the south side of the Dividend fault and the schist on the north. The outline of the stock is irregular, and extends about a mile from north to south and an equal distance along the fault. The only other porphyry outcrop of any size in the mining area is the Shattuck dike, which runs from above the Spray mine for more than a mile westward. Another important dike, the Sacramento, outcrops a few isolated places in the Naco limestone, southward from Sacramento Hill almost to the Briggs mine.

In the underground mine workings, the relative abundance of porphyry intrusions in the formations lower than the Carboniferous holds about the same as on surface. Besides the main mass of the Sacramento Hill intrusion there is another smaller, laccolithic mass of porphyry around the Lowell mine. This body is connected with Sacramento Hill and Sacramento dike by masses of porphyry, none of which reach the present surface, but which spread out mostly at the Carboniferous-Devonian horizon. This mass, with its accompanying contact breccia, is shown in Plate 6.

The mode of intrusion of the granite porphyry and its relation to the structure of the older rocks will be touched upon later. It will be enough to say here, however, that the intrusions took place in one general period, but not all at once, nor were the conditions the same at the beginning and end of the eruptions, or the composition of the rock exactly the same. Considerable alteration was also started in the rocks before the last porphyry was injected.

Lithology.—Although there is abundant fresh granite porphyry in the district most of the rocks around the ore zones have been so altered by mineralizing solutions that it is difficult to find material which represents the original granite porphyry of the main masses associated directly with the ores. Fortunately, the alteration processes have been so varied

that they have sometimes left untouched some of the phenocrysts, at other times the base, so that from various sources a reconstruction can be made of what was for a short time the rock of Sacramento Hill and the Lowell mass. Of the dikes fresh rock is available.

The pre-Cretaceous granite porphyry is, when fresh, a light-gray rock weathering to a light-red color on surface. It has abundant phenocrysts, up to ½ in. in diameter of feldspar, quartz and biotite, the relative amounts of these being variable. The base is a fine-grained mass, almost glassy at times and composing from one-half to about eight-tenths of the rock. The microscope shows that the rock varies from a typical granite porphyry to a rather basic monzonite porphyry, sometimes in the same sill or dike. The main masses were probably of the more acid variety, with well-developed quartz phenocrysts that have embayments due to reabsorption. There are no noticeable inclusions in the quartz. In the more basic varieties of the porphyry, quartz phenocrysts become extremely scarce.

The feldspar phenocrysts are mostly orthoclase with albite and oligoclase increasing as the rock becomes more basic. The orthoclase is in large crystals when it predominates and in small ones when plagioclases are abundant. The only intergrowth of the orthoclase seen is in the form of a micropegmatite with quartz, and at the rim of some large crystals in the dikes where an aureole of quartz and orthoclase usually forms around a pure orthoclase of quartz center. The plagioclases are all between albite and oligoclase and sometimes are in greater abundance and size than the orthoclase.

Biotite is the only ferromagnesian mineral of any importance and comes in large well-formed crystals as well as fine shreds. In most of the porphyry it has been altered before any other minerals, and its traces cannot be distinguished megascopically, but in some of the more basic varieties, which are also chloritized, biotite is the most conspicuous phenocryst mineral and gives a decidedly dark color to the rock.

The base of the porphyry is usually a cryptocrystalline mass of quartz and feldspar, found sometimes with a micropegmatitic intergrowth, at other times with incipient spherulitic growth and a few times with a holocrystalline but extremely fine crystallization.

As accessories, there are: apatite in well-formed needles sometimes noticeably abundant, and generally in the biotites; very few grains of pyrite, and dusty undeterminable inclusions in the feldspars.

The foregoing description of the unaltered porphyry is rarely applicable to the rock encountered in the mining area, as the porphyry may be found in the form of a white mass of sericite and quartz, or a dense dark-green rock composed mostly of serpentine and penninite, with all variations between. These characteristics will be considered later under metamorphism and mineralization.

Age.—The granite porphyry distinctly cuts the Pinal Schist, granite

and all the Paleozoic sediments, while pebbles of the altered sericitized and oxidized porphyry are found at the base of the Cretaceous Glance Conglomerate. It is, therefore, certain that the age of intrusion is post-Carboniferous and pre-Cretaceous. That the porphyry intrusion did not take place all at once, is shown by dikes of the slightly more basic varieties cutting the contact breccias of the more acid ones. This difference in age is further emphasized by differences in alteration, but, as will be shown later, there is no reason to believe that there was any great interval of time between the porphyry injections, but that they all belong to the same general period.

Rhyolite

This rock has not been distinctly recognized in the mining area of the Warren district, but due to its great similarity to the granite porphyry it may have been confused with the older rock, especially where both are found in dikes. The rhyolite from the mouth of Tombstone Canyon, whose field relations have already been given, is a gray to pink rock with decided flow structure in its surface forms. Phenocrysts are not abundant and are seen under the microscope to be entirely quartz and orthoclase, in an aphanitic or cryptocrystalline groundmass. Some of the dikes on Escabrosa ridge, and others northeastward from the mouth of Tombstone Canyon, have such a glassy groundmass, and so few phenocrysts of quartz and orthoclase, that they may very well be of the late rhyolite, especially as they are seen to cut other porphyritic dikes in places. The rhyolite of the Naco Hills described by Ransome is probably of the same age as that at the mouth of Tombstone Canyon.

Age.—Unfortunately no information is available at present for placing the age of the rhyolite any closer than post-granite porphyry. An erosion period separates the two, but whether it is the pre-Cretaceous or post-Cretaceous one cannot be definitely stated.

Other Dike Rocks

Hornblende Andesite.—At a few scattered places on the surface, and in the Shattuck, Cole, Wade Hampton and Wolverine mines, there are dikes of a dark-green, fine-grained rock, which is seen under the microscope to be composed of abundant small phenocrysts of andesine and labradorite and a few of hornblende, in a microcrystalline base of feldspar and fine magnetite grains. This andesite cuts all the pre-Cretaceous formations, including the granite porphyry, but its relation to the ores is not clear. The rock is considerably altered, and chloritic minerals especially penninite, as well as some pyrite, have been developed. In a specimen from the Wolverine shaft there is also some sericitization, so it is possible that the andesite may antedate the last of the mineralization period.

Diabase.—Olivine and augite diabase in small masses and dikes have

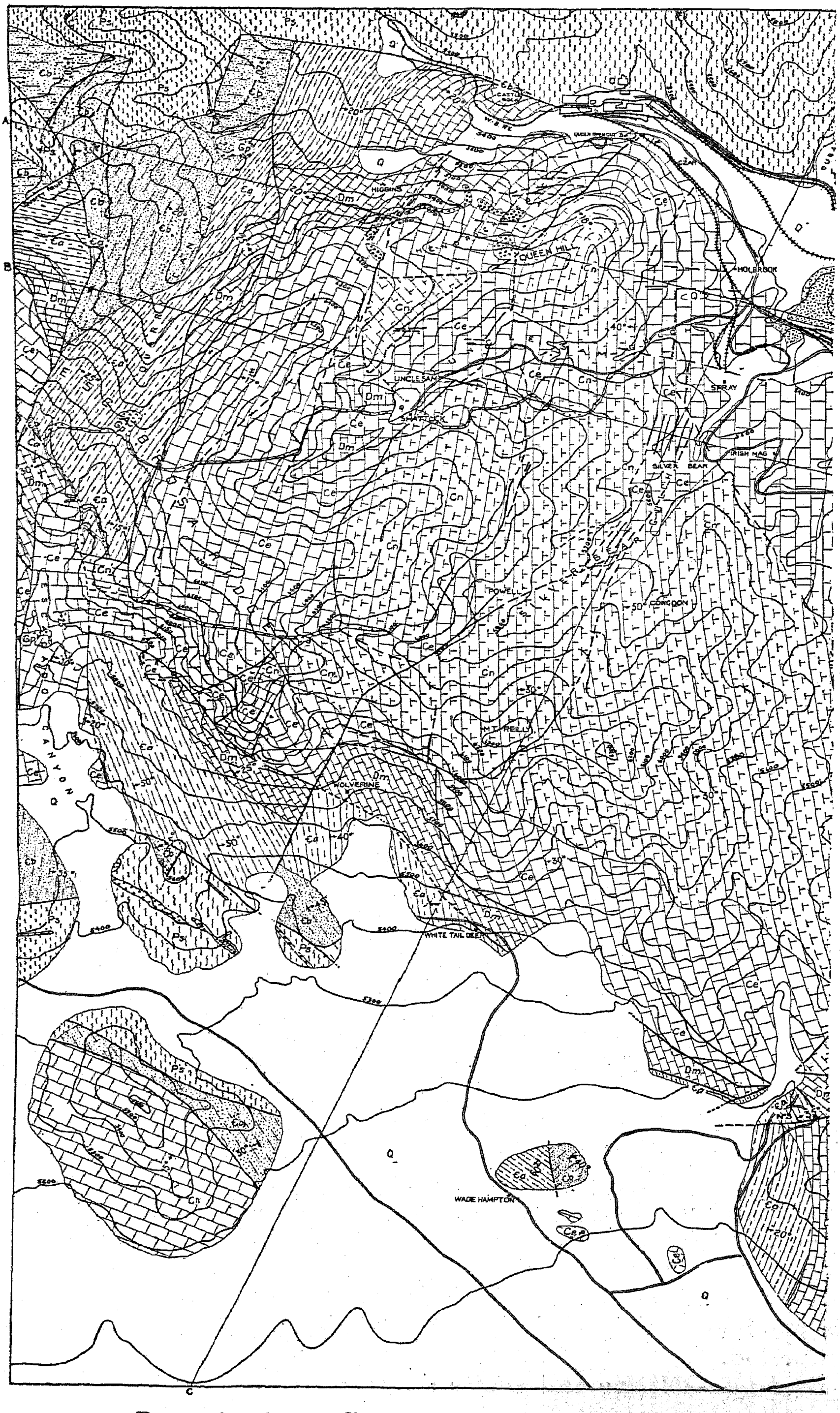
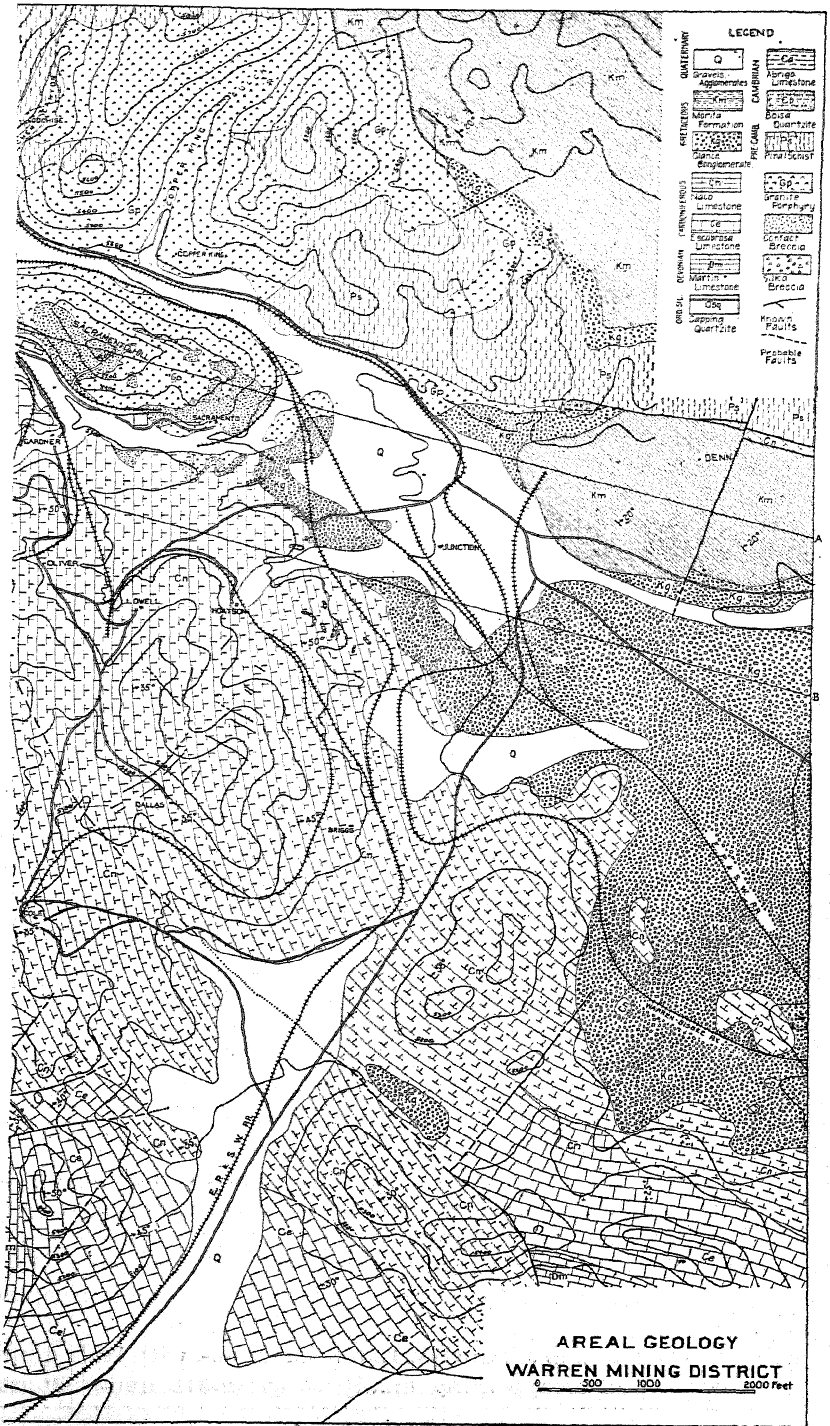


PLATE 4.—AREAL GEOLOGY MAP WITH SECTION LINES A–A, B–



AND C–C RELATING TO THE SECTIONS SHOWN BY PLATES 5 TO 7.

been found to belong to two distinct periods. The earlier is folded in with the Pinal Schist and is pre-Cambrian, while the latest cuts the Carboniferous limestones and is entirely unaltered. There is no known exposure of them underground.

Monzonite Porphyry.—The dikes of this rock mentioned by Ransome as occurring around the Glance mine have not been studied during the preparation of this report, and no similar rock has been found in the mining area, unless it is the one classified as andesite porphyry, or the basic porphyry found by Boutwell.

V. STRUCTURAL GEOLOGY

General Structure of Mule Mountains

Fault Blocks and Other Divisions

In this report the structural geology will be confined almost entirely to the southwestern part of the range covered by pre-Cretaceous sediments, as these play the most important part in the relation to the orebodies of the district. Reference to the areal geology map, Plate 4, and sections made along its lines *A–A*, *B–B*, and *C–C* shown in Plates 5, 6, and 7, will help make clear the following description:

As previously stated, the range is roughly divided by Tombstone Canyon and Mule Gulch into two parts. To the northeast, and forming about two-thirds of the range, Cretaceous sediments cover the whole area, dipping gently to the northeast and north, and being very little faulted. Where cut by the above-mentioned canyons, they overlie a nearly level plain of schist and granite. This area will be alluded to as the Cretaceous Tract.

The area southwest of the two canyons is almost entirely made up of much faulted and intruded schists and Paleozoic sediments, covered at the southeastern end by Cretaceous sediments overlying a rough topography of Paleozoic beds. It is this area which is studied in detail in this report.

Referring to the accompanying sketch map (Plate 1) of the range, it is seen that in a general way the dips of the Paleozoic beds radiate around the Juniper Flat granite mass. At the northwestern end they dip to the north and northwest. At Mt. Martin and Mt. Ballard, the dip swings to the west and southwest. From Mt. Martin to the southeast, the dips change from south to due east. Each change in dip and strike, however, is generally marked by a fault, the beds seeming to withstand folding to a remarkable degree.

In our division of this area into fault blocks, the divisions given by Ransome[7] have been generally followed, with a few minor exceptions and additions.

Starting at the north is the Bisbee Extension Block with beds dipping to the northwest, and disappearing finally to the north under Quarter-

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[7] *Loc. cit.*, p. 93.

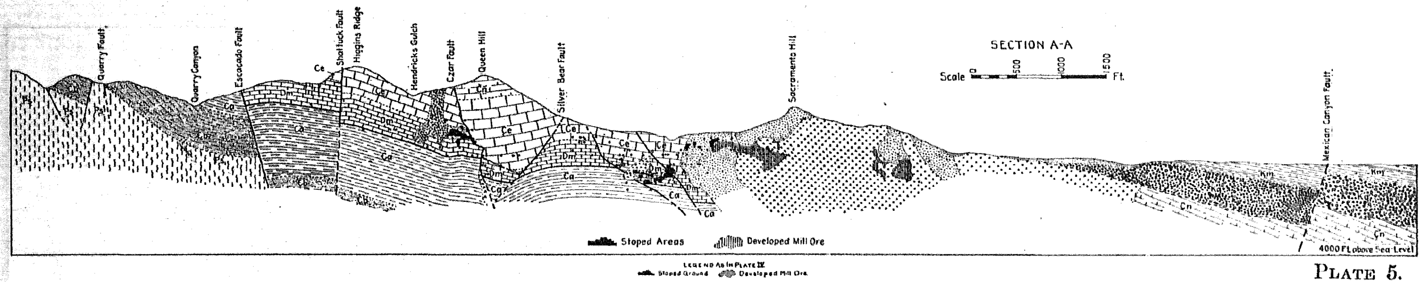


PLATE 5.

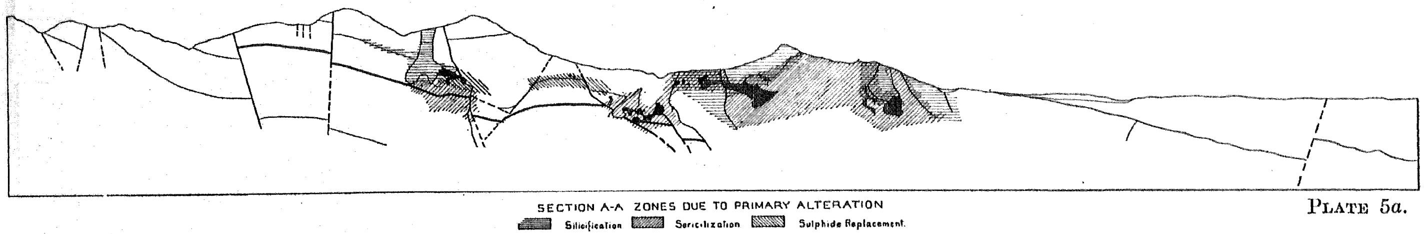


PLATE 5a.

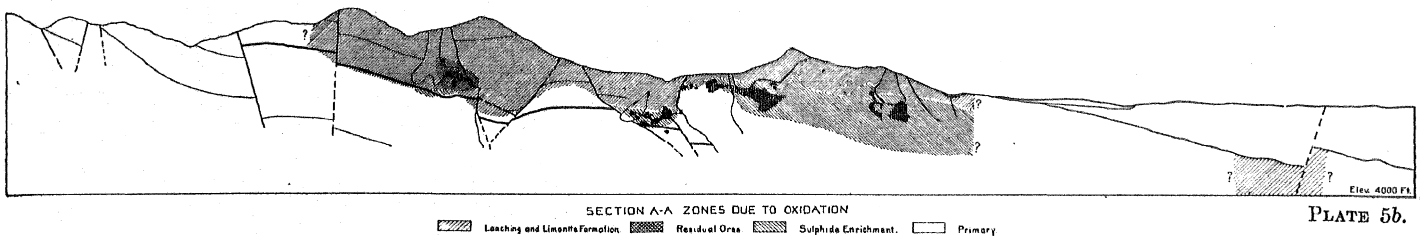
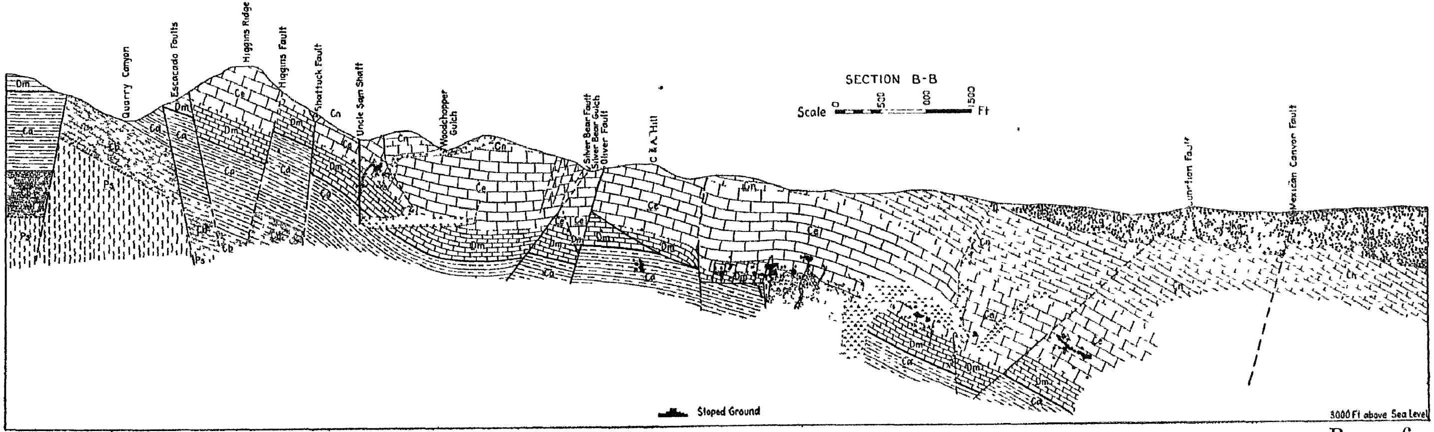


PLATE 5b.

PLATES 5, 5a, 5b.—GEOLOGICAL SECTION ALONG LINE A–A OF MAP, PLATE 4.



LEGEND AS IN PLATE IX

PLATE 6

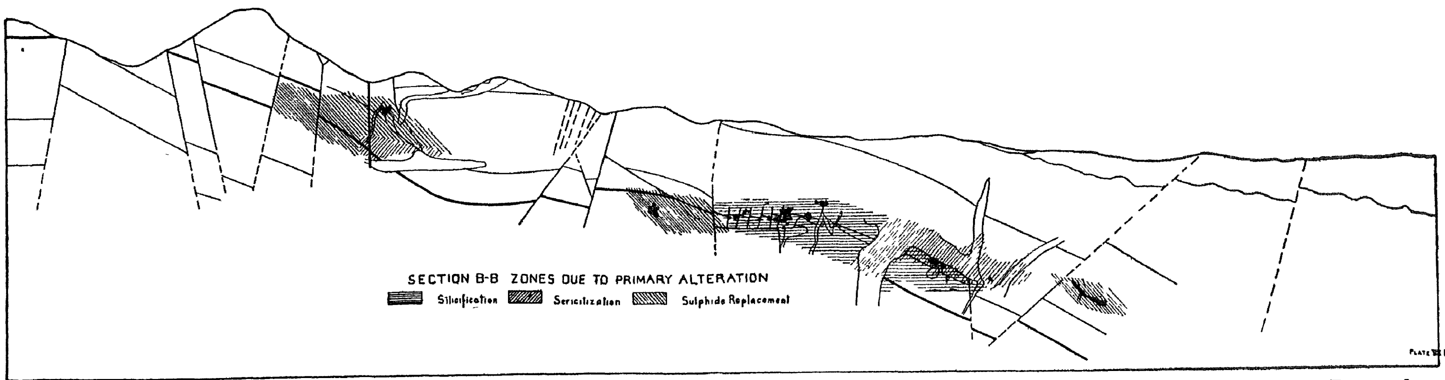
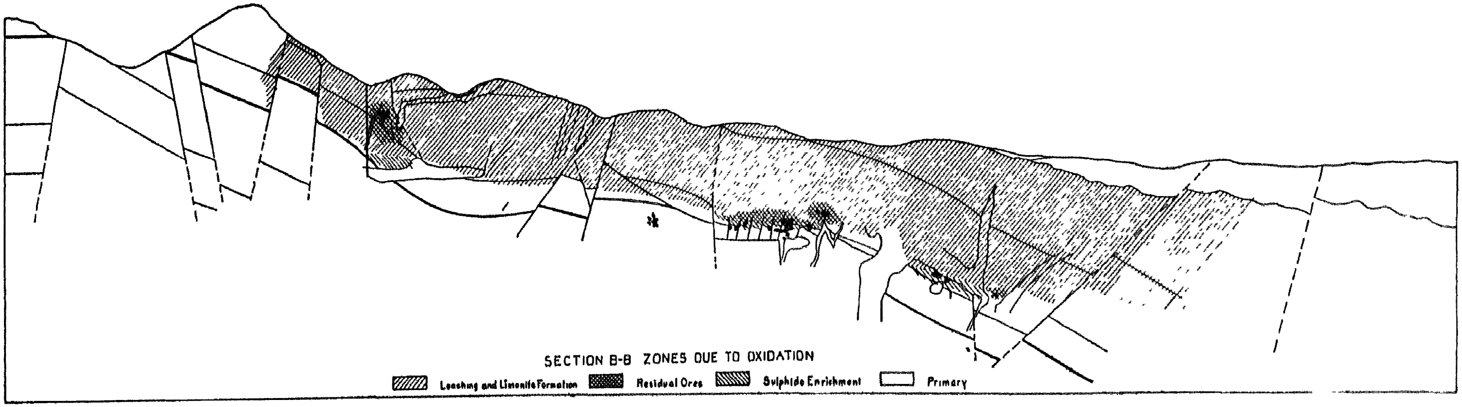


PLATE 6a.



Plates 6, 6*a*, 6*b*.—Geological Section Along Line B–B of Map, Plate 4.

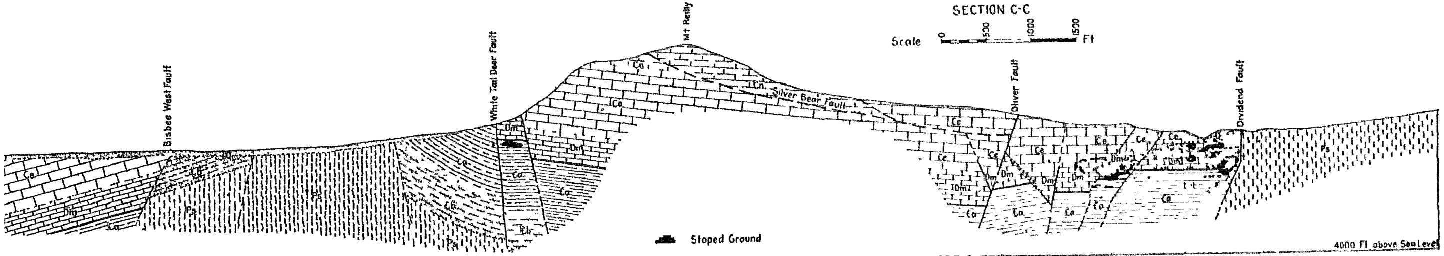


PLATE 7.

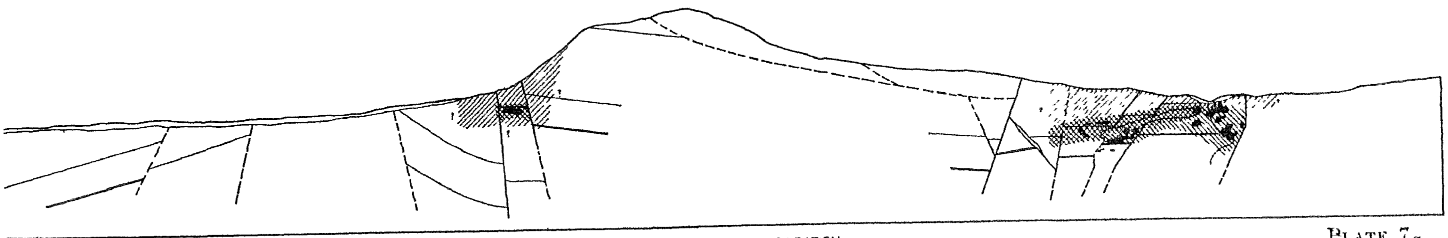


PLATE 7a

SECTION C-C ZONES DUE TO OXIDATION

Leaching and Limonite Formation · Residual Ores · Sulphide Enrichment · Primary

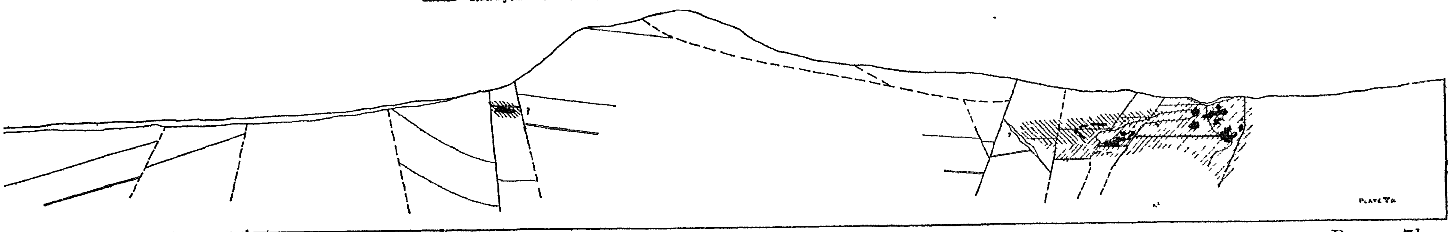


PLATE 7b.

SECTION C-C ZONES DUE TO PRIMARY ALTERATION

Silicification. · Sericitization · Sulphide Replacement

PLATES 7, 7a, 7b.—GEOLOGICAL SECTION ALONG LINE C–C OF MAP, PLATE 4.

nary wash. This block is bounded on the south by the Modern stockwork of granite porphyry, cutting through the schist and Paleozoic beds and striking northeast and southwest.

To the south of the Bisbee Extension Block are two blocks, the Escabrosa, extending from the Juniper Flat granite mass to the Escabrosa Ridge line of faults and porphyry intrusions, and the Bisbee West Block extending from Escabrosa Ridge to the southwest. The Escabrosa Block is up-thrown with respect to the Bisbee West Block, and consists almost entirely of schist and porphyry, until Mt. Martin is reached. Here we have the whole Paleozoic section exposed up to the middle of the Escabrosa limestone. The dips at Mt. Martin are southwest. The Bisbee West Block is covered at the northwest by Carboniferous beds dipping to the northwest and west. These are followed to the south by Cambrian beds, dipping to the west and southwest. The Escabrosa Block is bounded on the southeast by the Quarry Fault, beyond which are the Copper Queen and Don Luis Blocks. The Bisbee West Block is bounded on the south by the Abrigo Fault, to the south of which is the Naco Block; relatively very much down-thrown. To the northeast it is bounded by the Bisbee West Fault, to the north of which is the relatively up-thrown Don Luis Block.

The Copper Queen Block is down-thrown with respect to all the surrounding blocks. It is bounded on the north by the east and west Dividend Fault, which has a throw of at least 1,500 ft., throwing the Pinal Schist to the north against Paleozoic beds to the south. It forms a block with the Quarry Fault, a northeast to southwest fault which is the western boundary of the block. The Quarry Fault in turn, forms a block with the White-Tail Deer Fault which is nearly parallel to the Dividend Fault and throws Cambrian beds and Pinal Schist to the south against Devonian and Carboniferous beds to the north. The block is finally covered to the east by Cretaceous beds which are in turn covered by Quaternary wash. To the southeast, the block ends abruptly against the Gold Hill Overthrust Fault. The dips in the Copper Queen Block are generally to the east and northeast.

The Don Luis Block is a small up-thrown block between the Copper Queen and Bisbee West Blocks. The larger part of this block is covered by Quaternary wash, with numerous outcrops of schist and Cambrian beds. The dips in this block are variable but generally to the eastward.

In addition to these are four other blocks, three of which are evidently post-Cretaceous, and one probably so. These are the Tombstone Overthrust Block at the northwestern end of the range thrusting Naco limestone over Morita sandstone; the Naco Block bounded on the north by the Abrigo Fault; the Gold Hill Overthrust Block; and the Glance Overthrust Blocks at the southeastern end of the range.

The Cretaceous Tract, except where the sediments overlie the pre-

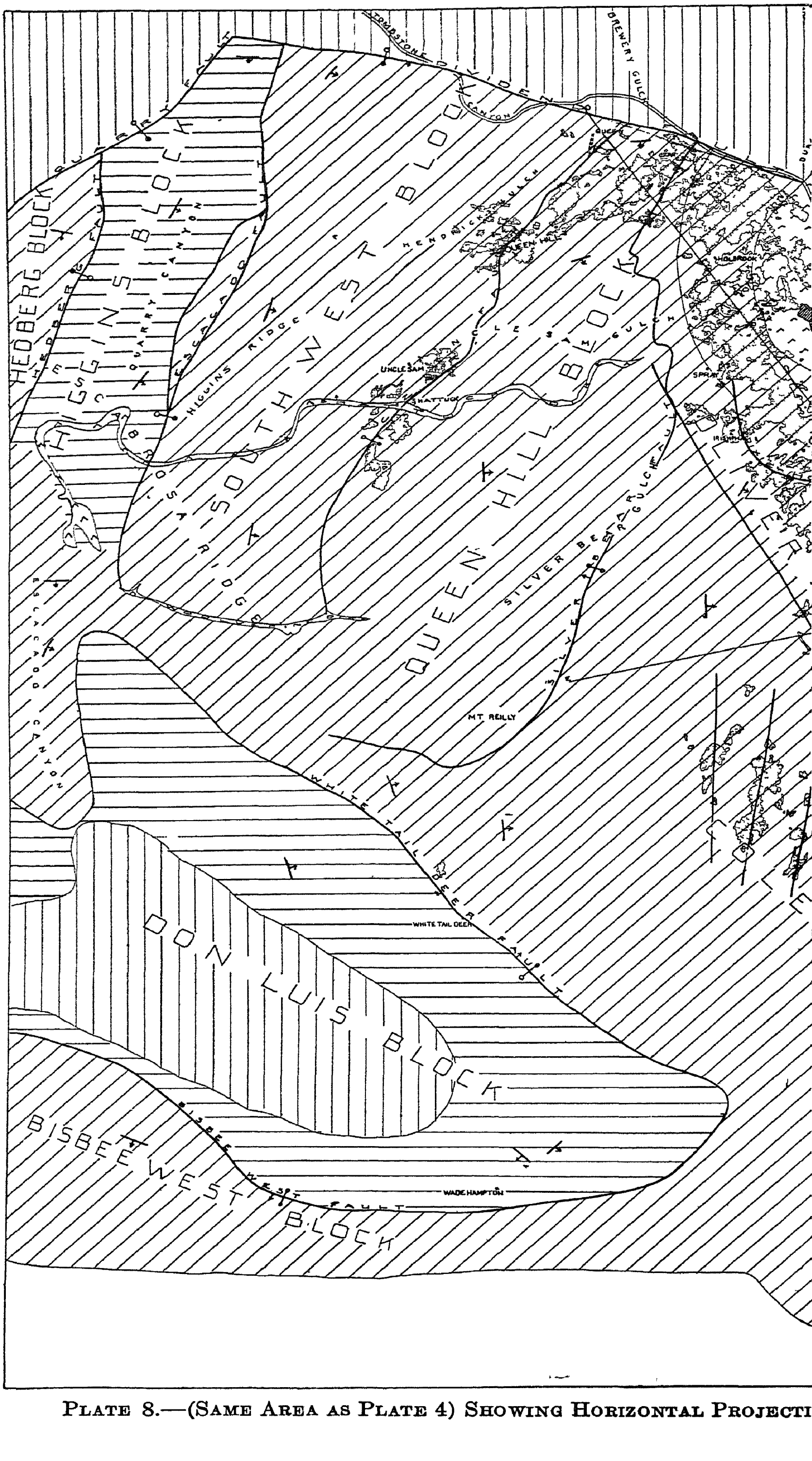
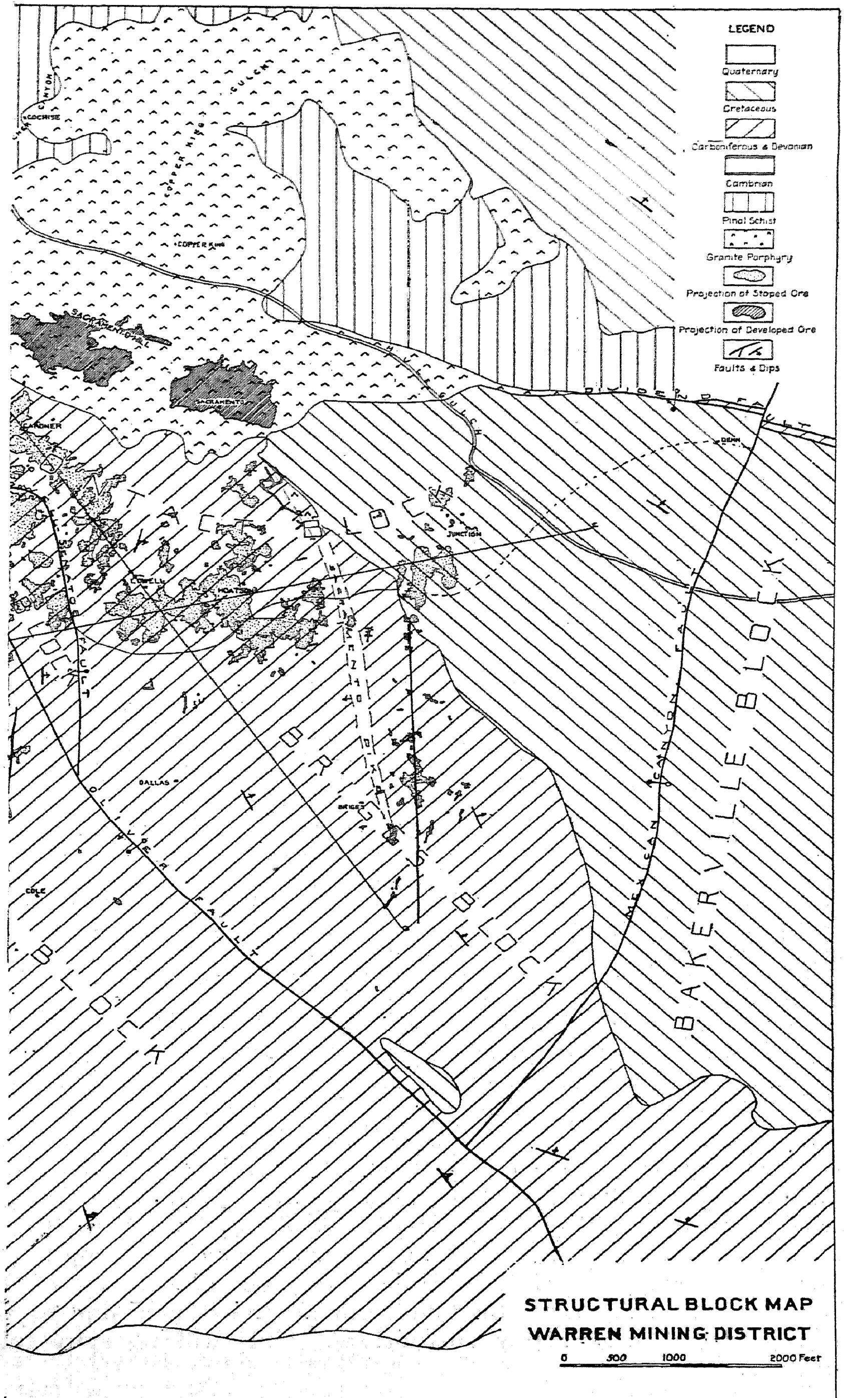


Plate 8.—(Same Area as Plate 4) Showing Horizontal Projection



OF STOPES AND SECTION LINES D–D AND E–E OF PLATES 9 AND 10.

Cambrian granite mass, and the Sacramento Hill porphyry, is devoid of igneous rocks. No intrusion has been observed cutting the Cretaceous over this area.

Igneous Rocks

The pre-Cambrian Juniper Flat granite mass has the form of a batholith. The contacts with the schist are clean cut. It probably forced its way up very slowly along some ancient line of weakness. There is no evidence of its having engulfed the schist, as no horses of schist are found inside the area exposed. From the nature of the schist, however, it is difficult to tell how it came in.

In all the pre-Cretaceous blocks mentioned above, extensive intrusion has taken place of granite porphyry, followed in some instances by extrusive rhyolites. These intrusions and extrusions came in for the most part along faults already in existence, but spread out in the less resistant sediments as sills and irregular masses. Although coming in at more or less the same time, some dikes clearly cut the others. The general habit of the porphyry was to follow some fault for a certain distance, and then cut across to another fault. The whole process suggests a very rapid intrusion.

In the Bisbee Extension Block, at the northwestern end of Escabrosa Ridge, there is an occurrence of extrusive rhyolite overlying an erosion surface of Naco limestone and an intrusive porphyry sill.

Cutting through the Sacramento Hill porphyry, north of Naco Road, are several small dikes of andesite. This same andesite occurs cutting Escabrosa limestones near the Cole shaft, and has been observed underground near the Shattuck, Wolverine and Wade-Hampton mines. At the Shattuck mine, it is undoubtedly later than the porphyry. This andesite in all observed cases has come in as very small dikes in already opened fissures. In only one case has an outcrop been observed in higher beds than the Martin.

In the post-Cretaceous blocks, no intrusions have been seen except in the Naco Block. Here, at the north end of the block, entering along the big Abrigo Fault, an extrusive rhyolite mass has come in. This mass is similar in all respects to the mass mentioned above in the Bisbee Extension Block. Except for this one case, the large faults of post-Cretaceous age have not been accompanied by igneous intrusion.

The most important, economically, of all these igneous rocks is the granite porphyry, as it alone seems to play an important role in the processes of ore deposition. By far the greater amount of porphyry in the district cuts through with clean contacts. The one exception to this rule is the mass of Sacramento Hill. The mode of intrusion of this particular mass is somewhat different from all the others, due to the accompaniment of mineralizing emanations. This will be more fully treated later.

Copper Queen Block

Fault Blocks

From an economic viewpoint, the Copper Queen and Don Luis Blocks are at the present time the only important ones in the range, and these two blocks, together with the mode of intrusion of the Sacramento Hill porphyry mass, will be more fully explained below.

The Copper Queen Block, whose details are best shown by Plate 8, lies in the heart of the range. It is a relatively down-thrown block, being bounded on the north by the Dividend Fault, a N. 75° W. fault dipping to the south from 60° to vertical. This fault has an estimated throw at the town of Bisbee of at least 1,500 ft. throwing Pinal Schist with a few outliers of Bolsa Quartzite on the north against Paleozoic sediments to the south. This fault apparently forms a block to the west with the Quarry Fault, a N.E.-S.W. fault which forms the western boundary of the area. To the south, the block is bounded by the White-Tail Deer Fault system parallel to the Dividend, throwing Bolsa quartzite and schist to the south against Abrigo and Martin limestones, to the north. To the east, the block is covered by Glance conglomerate and Morita sandstone, which are in turn covered by the Quaternary gravels of Sulphur Springs Valley. In the southeast, the block ends abruptly at the Gold Hill Overthrust Fault.

Starting somewhere near the Spray shaft, the block is cut by a N.W.-S.E. fault of considerable throw, the Oliver Fault, which dips about 60° S.W. This fault increases in magnitude toward the southeast and where it is finally covered by Quarternary wash, about halfway between Don Luis and the Country Club, it throws Abrigo limestone to the northeast against Naco limestone to the southwest, a throw of at least 1,500 ft. This fault is encountered in the Spray, Oliver and Cole workings. At the Spray the throw is about 200 ft.; at the Oliver it throws Abrigo limestone against Escabrosa with a throw of at least 500 ft. It finds topographic expression in the big draw between the Dallas and Cole shafts.

The Copper Queen Block, as seen by Plate 8, may be roughly divided into eight structural blocks. Starting from the west, we have first the Hedberg Block, relatively down-thrown, and with dips generally to the south. This is followed by the Higgins Block, up-thrown with dips to the east and southeast. Following this is the Southwest block, bounded on the east by the Czar fault system, and on the west by the Escacado fault system. This block is down-thrown relative to the Higgins Block but up-thrown in relation to the Queen Hill Block to the east. The dips here are nearly due east. The Queen Hill Block is a funnel-shaped block of Naco and Escabrosa limestone. It is well shown in section A-A, Plate 5. It is bounded on the west by the Czar fault system, and on the east by the Silver Bear Fault. The dips on the surface are irregular

but, in general, to the east. Underground, below where the two faults wedge out, the dips are consistently east. Surrounding Sacramento Hill, to the south and southwest is a dropped block, which we may call the Contact Block. It is bounded on the north and northeast by the Dividend Fault and Sacramento Hill, and on the south and southwest by the Dixie-Howell-Senator Faults. To the southwest of this Contact Block and up to the Oliver Fault is the Oliver Block of relatively up-thrown beds. Southwest of the Oliver Fault is the Cole Block and northeast of the same fault is the Briggs Block which extends east to the post-Cretaceous Mexican Canyon Fault. The dips of all these blocks more or less surrounding Sacramento Hill are nearly due east. The Oliver Fault has tended, however, to swing the up-thrown side, especially where the throw becomes greatest, around to the northeast as is shown in the detailed geological map of the district, Plate 4.

To the east of all these blocks is an eighth block, the Bakerville Block, almost entirely covered by Glance Conglomerate. This block is bounded on the west by the Mexican Canyon Fault, striking about N. 15° to 30° east and dipping to the northwest. This fault is a late one and extends beyond the Dividend Fault, causing the prominent dislocations of the Mural Limestone seen north of Lowell and Warren.

As is shown in section B-B, Plate 6, the apparent dip of the beds in the whole block is very much less than the actual. This is due to minor faulting or slipping which has had the effect of dropping the beds very consistently to the west and raising them to the east. This is one of the typical structural features of the camp and seems to be caused by the extreme resistance of the beds to folding.

There are two ages of faulting, one pre-Cretaceous and the other post-Cretaceous. As has previously been shown, the porphyry in the district is pre-Cretaceous. The age of the faulting has therefore been determined in large part relative to the age of intrusion. The only large fault which is definitely post-Cretaceous is the Mexican Canyon Fault. As the fault apparently blocks with the Oliver Fault as shown in Plate 8, the Oliver is probably also post-Cretaceous. This is borne out by the character of the exposures underground. There, practically no mineralization accompanies it, and in Oliver ground it apparently cuts off an orebody, which as will be seen later, would make it post-Cretaceous. Both of these faults have the same effect, that is to raise the sediments to the east, and to drop them to the west. The probability is that the Junction Fault, shown in section B-B, Plate 6, is also post-Cretaceous, as it is reported to cut off certain parts of the Junction orebodies, and it is also possible that a large part of the north to south slipping in Lowell and Gardner ground is also post-Cretaceous. Here the throw of each individual fault is so slight, and the ground so highly metamorphosed, that it is not susceptible of proof one way or another.

All other faults, so far as observed, have had their major movements in pre-Cretaceous time and before the introduction of the porphyry. Some faults have had movements in both ages. The Dividend shows some movement at the present time, and enough movement occurred in post-Cretaceous time to throw out the Glance Conglomerate. Some post-intrusion slipping has also undoubtedly taken place along the Czar Fault.

Intrusions

As has previously been shown, the block has been extensively intruded by granite porphyry. The largest and most important mass is that of Sacramento Hill, the other occurrences being more or less radial offshoots of dikes and sills from this central mass. It has also been shown that underground exposures below the Escabrosa horizon are much greater than surface exposures.

The path of the Sacramento Hill porphyry was the previously opened-up Dividend Fault. Underground development has shown that the porphyry acted as a plug, and that none of the later movements on the fault have taken place along the original line through the mass. The offshoots of the mass have generally followed major lines of weakness such as the Czar, and Silver Bear Faults. Some, however, such as the Shattuck dike, have cut across the country irrespective of fault lines.

The Sacramento Hill, and the connecting Lowell, mass differ from all the other porphyry occurrences in the range, in that they were accompanied by mineralizing emanations, which have completely altered them. These emanations, besides altering the porphyry itself, changed and broke up the surrounding sediments. In consequence of this, as the mass worked its way up, it dragged in pieces of schist, quartzite and limestone along its edges. The final result was a central core of metamorphosed porphyry, with a periphery of contact breccia, made up of highly silicified and altered fragments, sometimes rounded and sometimes angular, of schist, porphyry, quartzite and metamorphic limestones. This contact-breccia zone has a variable width, with a maximum of about 1,000 ft. It has fairly sharp contacts with the surrounding limestone, but tends to grade off into brecciated porphyry, and finally blocky unbroken porphyry, toward the center of the intrusive mass.

Don Luis Block

The Don Luis Block is an elliptical-shaped block up-thrown relative to the Copper Queen. It is bounded on the north and east by the White-Tail Deer fault system, throwing Abrigo and Martin limestone against schist, Bolsa quartzite, and some Abrigo limestone. To the south it is bounded by the Bisbee West Fault, which is covered by the Quaternary

wash toward the east and toward the west it is bounded by the Quarry Fault. The dips are variable, but for the most part are to the northeast and east.

Intrusions and Orebodies

The block is intruded both by granite porphyry and by andesite, the latter, however, not outcropping. There are two ore occurrences in the block, one at the northern boundary, and rightfully in the Copper Queen Block, and one at the southeastern end. The first of these, the White-Tail Deer orebody, occurs as a replacement of Abrigo limestone, about 50 ft. from the base of the Martin. It is not associated with any known intrusion, but is close to the White-Tail Deer fault system. The second occurrence is that of the Wade-Hampton. Here the ore occurs as lead and copper ore in a fault separating Abrigo limestone from Bolsa quartzite. The ore here is apparently associated with a dike of andesite and occurs in the crushed Abrigo limestone on the fault.

VI. GEOLOGIC HISTORY

The earliest geologic age is recorded in the district by the pre-Cambrian Pinal Schist. As has been previously stated, this was probably derived from arenaceous sediments.

Due to the extreme metamorphism, the geologic history subsequent to the deposition of these early sediments, and up to the deposition of the Cambrian sediments, is obscure. The sediments were probably subjected to extreme folding, intruded by diabase, and deeply buried. After the metamorphism was complete, the schist was intruded by a large mass of granite. From then on to Cambrian time, erosion worked on the whole complex, wearing it down to an even peneplain.

At the beginning of Cambrian time, a gentle subsidence took place, resulting in shore-line deposits of quartzites, the subsidence becoming more rapid toward the end of the period, in which time were deposited finer sands and some shales. The land surface toward the end of the Cambrian period became more distant, as evidenced by the deposition of the Abrigo shales, and shaly limestones.

At the end of Cambrian time, a gentle rise took place over an extensive area, bringing the flat sea bottom close to the surface, with land actually emerging probably toward the south. During the Ordovician and Silurian ages, this lowland surface of Cambrian beds was subjected to slow erosion simultaneously with the deposition in the district of an 8-ft. bed of quartzite. A slow subsidence then took place to the north of the district, allowing the thicker deposits of quartzite in the Yellowstone Range of 40 ft., and at Globe of 500 to 700 ft.

At the end of the Silurian, a sudden subsidence took place, resulting

in deeper-sea conditions and thus the Devonian age is represented by fairly pure dolomitic limestones.

The sea evidently continued to deepen all through Mississippian time with the deposition of the pure Escabrosa limestone. During Pennsylvanian time, subsidence did not keep pace with deposition and the sediments became increasingly more shaly and cherty, showing a nearer approach of land surface and shallower seas. At the end of the Pennsylvanian, and probably during the Permian age, a violent uplift took place, resulting in the doming of the sediments around the pre-Cambrian granite mass. The sediments were shattered by extensive faults, the major lines of weakness being northwest and southeast. The granite mass itself resisted this shattering to a great degree except around its edges. Immediately following the faulting, or partly accompanying it, granite porphyry was intruded, following generally the major lines of weakness, but also shooting sills and dikes into the less resistant sediments. Accompanying one of these porphyry intrusions, that of Sacramento Hill, were mineralizing emanations, which metamorphosed the surrounding sediments and the porphyry itself, and were the source of our present orebodies.

During Jurassic and Triassic time, the whole area was subjected to erosion, during which time most of the upper Carboniferous beds were stripped off, and at the crest of the dome, around the granite and schist of Juniper Flat, the old pre-Cambrian complex was laid bare. During this erosional period, the orebodies were subjected to oxidation, which would place the age of primary ore formation definitely at or near the age of the uplift.

At the end of this period, a remarkably sudden subsidence took place, before any of the land except the top of the dome had been leveled off, and the rough topography surrounding was filled up with the extremely coarse Glance Conglomerate. The subsidence was so sudden that the fragments making up this conglomerate were hardly worked over at all, it being made up of angular boulders, some as large as a yard in diameter, of schist, quartzite and limestone. The roughness of this old topography is well shown in the basin covered by conglomerate, in which the town of Warren is situated.

After the whole country was leveled off by the Glance Conglomerate, the subsidence became less rapid, and shore-line deposits of sandstones and sandy shales were laid down, followed by deeper-sea conditions, during which the pure Mural Limestone was deposited, showing abundant Cretaceous (Comanche) fossils. Shore-line deposits followed this again to the end of Cretaceous time.

During Tertiary time, a second uplift took place, along almost the same axis as the previous one. This also took the form of a dome around the old granite mass, and the Escabrosa Ridge porphyry intrusion. This uplift, however, was not as violent. It was accompanied by extensive

block faulting, with little disturbances of the beds within the blocks. The major faults took the form of overthrusts, the largest being the Tombstone and Gold Hill faults. Either accompanying the uplift or subsequent to it, some intrusion of monzonite porphyry took place, at the southern end of the range. Probably some of the extrusive rhyolite also belongs to this age of intrusion. Finally the whole range was subjected to a tilt to the northeast of about 15°.

From then on to the present time, erosion has been steadily at work, and has again laid bare the old pre-Cretaceous sediments along the crest of the dome, with Cretaceous beds exposed to the northeast. To the southwest, the Cretaceous has been all eroded, due to the final tilting. The only evidence left of the second doming is found at the southern end of the range, as shown by Plate 1. During this last erosion period the orebodies have been further subjected to oxidizing and enriching processes, so that we owe their condition as we find them today, to two widely separated periods, in each of which the original primary ores were worked over by surface waters.

VII. METAMORPHISM AND MINERALIZATION

General Metamorphism

In a region that has been subjected to as many geological changes as the Warren district, it was to be expected that the rocks would show many variations in texture and mineralogy from the unaltered types. The effects of regional metamorphism as well as dynamic are represented in the Pinal Schist as has been shown in describing the lithological characters of this formation.

The Cambrian formations have also been changed throughout their occurrence in the district by a process related to regional metamorphism in the wide scope of its action. The induration of the pure Bolsa quartzite has gone on until it is of glassy texture, but in the clayey, magnesian Abrigo limestone, besides the segregation of chert, the impurities have everywhere combined into such minerals as epidote, zoisite, chlorite, serpentine, and possibly albite. No such action is at all observed in the conformably overlying Martin limestone, which strongly suggests that the changes took place in the long period separating the middle Cambrian from the Devonian. But since field evidence excludes any great disturbance during the unrepresented period, it is probable that the general metamorphism in the Abrigo was caused by the deep burial under more than 4,000 ft. of overlying limestones after the impurities in the beds had had a chance to segregate and rearrange themselves due to the slow action of cold surface solutions during Ordovician-Silurian times. The necessary temperature conditions were probably obtained by the rise in iso-

therms due to deep burial and to the post-Carboniferous intrusive period when a general rise in temperature could effect changes in the more susceptible impure sediments.

Whether this tentative explanation is correct or not, the importance of recognizing the general distribution in the Abrigo of minerals which may be mistaken for indications of contact metamorphism cannot be overemphasized. This formation is of considerable economic importance and the underground search for ores cannot be aided by the same indications as in the higher limestones.

Contact and Hydrothermal Metamorphism

That the granite-porphyry intrusion was the supreme factor in the ore deposition of the district cannot be doubted in view of the intimate association of the ores with the porphyry and the intense changes that have taken place in the rocks around the main intrusions.

It has been customary to separate the metamorphism of the rocks around a contact into contact metamorphism, caused by the igneous rock itself, and hydrothermal metamorphism, caused by the heated solutions emanating from the cooling magma. This distinction will not be made in the present article, because there has been nothing found by the authors to indicate that there is any genetic difference in the causes for changes in the rocks, outside of differences in intensity or quantity. The differences in *effects* are not to be confused with the causes, as the first are subject to the many variations of the rocks encountered, their physical and chemical differences, and their previous alteration. The metamorphosing agencies vary also with the distance from their source in the effect they may have on the same rock, but all changes observed so far have usually been graded and not susceptible to sharp definition.

In general, it may be stated that the two main centers of porphyritic intrusion are also the centers of metamorphism. Around Sacramento Hill the effects of metamorphism can be seen on the surface, but around the Lowell center the alteration did not extend through the covering of upper Carboniferous limestone. The effects around Sacramento Hill are more pronounced on the side of the calcareous sediments, which have been so much more altered than the schist of the north side. The porphyry on the south, as well as the covering remnants of contact breccia, are highly silicified and stand out as the crest of Sacramento Hill. The topographical depression around this hill is due to less silicified porphyry breaking through the contact breccia where this last turns from a more or less vertical to a horizontal body. Outside of this moat-like depression is a ring formed by the highly silicified contact breccia which grades into silicified metamorphic limestone, and this into limestone with a decreasing amount of metamorphic minerals until a belt of marbleized rock is

reached, from 200 to 1,000 ft. away from the porphyry. The zone of recrystallization also fades outward irregularly to the unaltered sediments.

On the north, or schist side of the contact, there is some silicification of the older rock, accompanied by considerable pyritization, but this very soon fades out into the unaltered schist.

Underground, in tracing out the zones of rock alteration due to the intrusion and its accompanying solutions, it has been found that they roughly correspond to the ones just mentioned as appearing on surface, with the following general modifications: The zones extend farther than the general surface contours along extensions of the porphyry mass, especially the zone in Lowell ground, and along lines of fracturing which may or may not also be lines of faulting. The extent of the zones is remarkably influenced by the formation they are traced in, the changes going farthest at the top of the Devonian and the top of the Cambrian. Local variations are found to break the arrangement of the zones of alteration where there are minor centers of strong action, such as close to the Shattuck dike in the Uncle Sam and Shattuck ground, in the Southwest mine, and in several other places.

The Effect of Contact and Hydrothermal Metamorphism on the Formations

Before taking up the effects of contact and hydrothermal alteration on the rocks of the district it is well to say that hydrous metamorphism or oxidation has obscured or complicated in a great measure the results of contact metamorphism, and in many cases made the exact separation of the process involved too difficult and useless to be undertaken in the course of economic work.

The changes due to circulation of meteoric waters will be taken up later. The mine workings have now gone down far enough in all zones to disclose the fact that there is a depth below which the rocks as well as the ores are different from those closer to the surface. Here there are none of the minerals evidently due to the process of oxidation, and mineralogical associations continue the same indefinitely downward, without the vertical variations of the upper portions. This is the criterion that has been used in separating the effects of primary and secondary processes in the rocks and in the ores. Though not found in this order in the mines, primary effects will be considered first.

As a whole the impression given by the rock alterations in the Warren district is one of abundance and persistence rather than great intensity. There are no high temperature minerals developed in any great amount, or if they were developed at some stage, they have now been replaced. Garnet, diopside, wollastonite, scapolite, and vesuvianite have been observed, but in very small amount, and never forming an important part of a formation. Tremolite, actinolite, and edenite are far more common.

The distinguishing minerals of alteration zones are quartz, sericite, chlorites, especially penninite, and the oxides of iron, magnetite and specularite, as well as the metallic sulphides of iron, copper, zinc and lead.

Metamorphism in Contact Breccia

Around any of the granite porphyry that has been accompanied by mineralizing emanations, such as that of Sacramento Hill, the contact breccia is marked generally by an extreme amount of silica which replaces all other gangue and rock-forming minerals. Pyrite and sericite are next in abundance, with chloritic minerals in variable amounts and calcite practically unknown. Schist and quartzite fragments have naturally remained the least altered in this mass and are consequently recognized most easily. In this contact breccia, or at the edges, there are bodies of intergrown magnetite, hematite and pyrite, with associations of the best-formed garnet, wollastonite, and other contact minerals. Silicification usually decreases toward the outer edge of the breccia, sericite and chlorites increasing. In many thin sections studied, the prevailing impression gathered is that in most of the breccia silicification is the last process involved, following sericitization and chloritization.

There are, however, distinctly later porphyry dikes cutting the breccia, as has already been stated, which have also been accompanied by metamorphosing emanations, the result being a complication and confusion of the processes involved. Certain highly chloritized portions of the breccia, where penninite replaces quartz in a noticeable way, are the result of these conditions, which are economically important, since copper minerals are concentrated in very pure form by the superimposed alteration. There are also other observed forms of this repeated alteration depending on the variations of either the first or the last. But it is generally true that the second process never reaches the point of adding silica or even adding sericite to the previously changed rocks.

Metamorphism in the Contact Limestones

The transition from breccia to limestone that can be recognized is apt to be very sudden in the Carboniferous horizons, and graded in those below. This is because the alteration of the lower limestones is more pronounced. The impure, and at the same time easily crushed, Devonian and Cambrian formations become a mass of sericite, penninite, calcite, and quartz, with some tremolite, garnet, diopside, wollastonite, and epidote, while the Escabrosa limestone usually has just a narrow fringe or a few bands of metamorphic minerals and the mass of the rock is simply recrystallized to marble. As extremes of alteration in this contact zone there are places where the limestone, especially the Devonian, is converted into a highly sericitic or chloritic mass, or into almost pure quartz.

There is some evidence in this zone also that silicification is the last process affecting the rocks under ordinary conditions.

Farther from the contacts or the centers of mineralization the changes visible in the limestone are a decrease in silica and sericite in all the rock, while these minerals may persist in breaks and joints. Recrystallization of calcite, but not of the impurities, is very common.

In this contact zone the accumulation of sulphides is marked by the increase in either sericite, silica, or chloritic minerals around the sulphides.

Metamorphism Outside of Contact Zones

Sometimes rocks with considerable alteration are encountered far from any known porphyry, in which case it may be that solutions have traveled along fractures, or that they may have come with undiscovered porphyry. Usually, however, the copper ores have been found replacing the limestone very much farther away than any general alteration of the formations, in which case the transition from fairly unaltered rock to sulphides is sudden, but bears some relation to the local structure.

Metamorphism of the Porphyry

The primary solutions emanating from the granite porphyry affect that porphyry in about the same way as they do the adjacent rocks. If the nearby rocks, whatever they may be, are silicified, the porphyry will be also. If sericitized or chloritized, the same processes will have affected the igneous rock. The main difference comes in the relative amount of sulphides which have formed during these changes. Of course, the same minerals will not form in limestone as in igneous rock, but an increase or decrease in silica or magnesia or iron will be parallel in both cases.

The main mass of Sacramento Hill is so highly altered that a thin section of this rock shows only a mat of sericite plates, with variable amounts of quartz, pyrite, and chlorite, without much being left to distinguish between quartz, feldspar and mica phenocrysts, and the base. Close to the breccia, silicification is common, accompanied by disseminated copper deposits.

In the dikes radiating from the hill, the changes in alteration of the porphyry are very clearly marked, as well as the relative ages of succession of the processes. Receding from the silicified portions, sericite increases and the outlines of phenocrysts, first of quartz, then of feldspar, begin to appear. Lastly appear the outlines of biotite, marked generally by successive laminæ of sericite and pyrite. Farther away penninite is found in increasing amount, the quartz phenocrysts being unattacked and some of the plagioclase feldspars scarcely altered. The chloritic minerals appear as remnants replaced by sericite.

When sericite has almost all disappeared, and there is no silicification, epidote, zoisite and calcite are sometimes found as alteration or meta-

morphic minerals in the porphyry. At this zone there are also found portions of the rock converted into almost pure penninite and edenite, with some serpentine, as if the ferromagnesian minerals lacking in the more sericitic rock had been driven to accumulate and replace other portions. Only the presence of some unaltered quartz phenocrysts and very resistant apatite crystals helps to determine the original rock, although transitions have been exposed in some of the workings.

Sulphides in some amount invariably accompany the contacts of porphyry that is altered to this degree.

Farther away from the centers of action slight chloritization of the base, of some feldspars and of micas, and eventually of the micas alone, is the only alteration noticeable. This fades out in a few observed cases to fresh rock, or rock affected only by meteoric waters. The contacts of this less altered porphyry have either just a little pyrite and sphalerite, or no sulphides at all, and even have had no appreciable effect on the immediately adjacent limestone when seen in thin section.

The alteration of the more basic dikes which cut the contact breccia is also similar to the one induced in the breccia, except that the earlier changes are not observed in these dikes.

Summary

To summarize, it may be said that the weakest as well as the farthest reaching alteration in the porphyry is chloritization. Next comes epidotization, which is of minor importance. Sericite and finally quartz are the additional minerals formed with the increase in the strength as well, probably, as the temperature of the emanations. The minerals formed by the stronger solutions are formed by replacement of the earlier ones, in the ordinary case when metamorphism is advancing in intensity from the center. Superimposed alteration may be caused by later intrusives, or in certain instances by the decrease in intensity in the dying-out period of alteration.

Some unaltered dikes in Holbrook and Czar ground are found in zones where the rest of the rocks show intense sericitization, including some masses of porphyry. These dikes are associated also with some sulphide ores, the contacts being far more distinct than ordinarily. If it were not for the slight sericitization of these dikes close to the ore, while away from it they show no hydrothermal action, it would be possible to believe that they are the result of post-mineralization intrusions.

The Shattuck dike shows similar phenomena, as it is very fresh for long stretches, and then, close to the ore zones in Uncle Sam and Shattuck ground is almost as altered as any rock around Sacramento Hill, showing that the altering solutions did not accompany all the porphyry but found their way through special channels.

The following partial analyses of altered porphyries are typical:

	I	II	III
Au, ounces	*Nil*	*Nil*	*Nil*
Ag, ounces	0.04	0.08	0.04
	Per Cent.	Per Cent.	Per Cent.
Cu	0.24	0.11	0.11
Pb	Tr.	Tr.	Tr.
SiO_2	55.9	76.6	73.3
Fe	9.8	5.6	5.7
CaO	1.8	3.1	0.5
MgO	0.5	3.2	2.8
Al_2O_3	16.6	7.4	10.1
S	8.8	1.0	0.4
Mn	0.6	0.6	0.6
Zn	Tr.	Tr.	Tr.
P	0.08	0.10	0.09
Totals	94.32	97.71	93.60

I. Highly sericitized porphyry with pyrite, from Sacramento dike, close to Sacramento Hill.

II. Chloritized, partly sericitized and epidotized porphyry, from Sacramento dike, about 1,000 ft. away from the hill.

III. Slightly chloritized porphyry, Dallas mine.

The general zones of primary alteration are shown for the sections through the camp in Plates 5*A*, 6*A*, and 7*A*. Silicification and sericitization have been explained already. Chloritization is not marked because it is too irregular, and though important in the porphyry, the sediments in that zone are not generally affected. The zone of sulphide replacement was introduced, where the limestones surrounding sulphide replacements were affected so slightly immediately away from the orebodies that there can be said to be practically no metamorphism.

VIII. THE ORES OF THE DISTRICT

General Occurrences

The general occurrence of the orebodies can be best appreciated by reference to the horizontal and vertical projections of the stopes given in Plates 8 to 10.

In the horizontal projection (Plate 8), there is a well-marked crescent-shaped zone around Sacramento Hill, with an extension around the Lowell mass of granite porphyry. Along the Dividend and Czar zones the effects of major structural lines are well marked by the abundance of ore taken out. The Shattuck dike-Czar zone intersection has its group of orebodies, and extensions and outliers from the general zone follow both the Sacramento dike and the series of breaks in Briggs and Cole ground to the southward.

In the vertical projections (Plates 9 and 10) the inclination of the ore zone is shown; also in Plates 5 and 6 the stratigraphical position is seen to follow in general the dip of the limestones, at horizons varying from the top 300 ft. of the Cambrian to the lower 300 ft. of the Carboniferous.

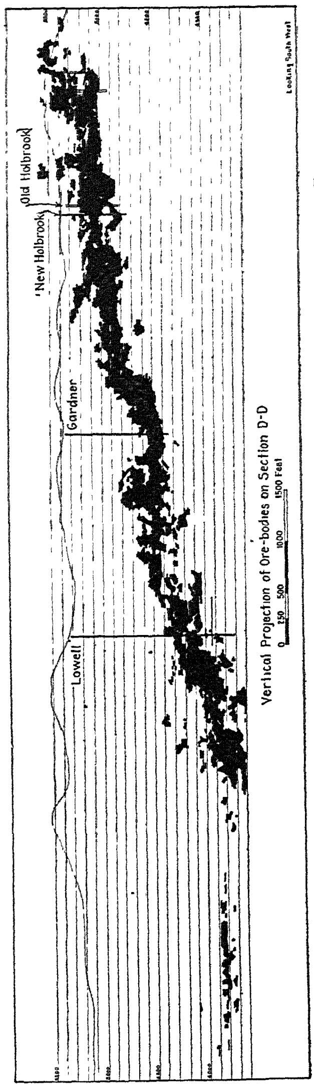


PLATE 9, SECTION D-D.—SHOWING INCLINATION OF ORE ZONE FOLLOWING DIP OF LIMESTONES.

CLASSIFICATION OF THE OREBODIES

As in the case of the metamorphism and alteration of the rocks, the authors have found no reason to separate the orebodies due to genetic differences. All the ores are believed to be due to one general period of

primary mineralization which followed closely the intrusion of the main porphyry masses. Differences in the ores are ascribed mainly to the character of the formation that they replace and to the amount of general

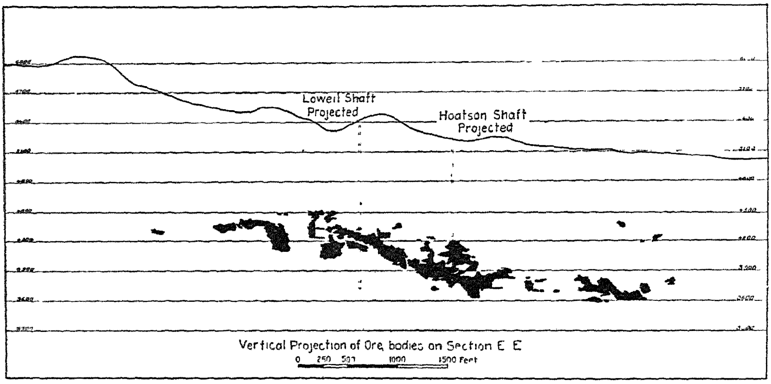


Plate 10, Section E–E.—Ore Zone Between Lowell and Hoatson Shafts Following Dip of Limestone.

metamorphism in the surrounding rocks due to the distance from the centers of alteration. All the ores are also believed to be formed by metasomatic replacements of various rocks.

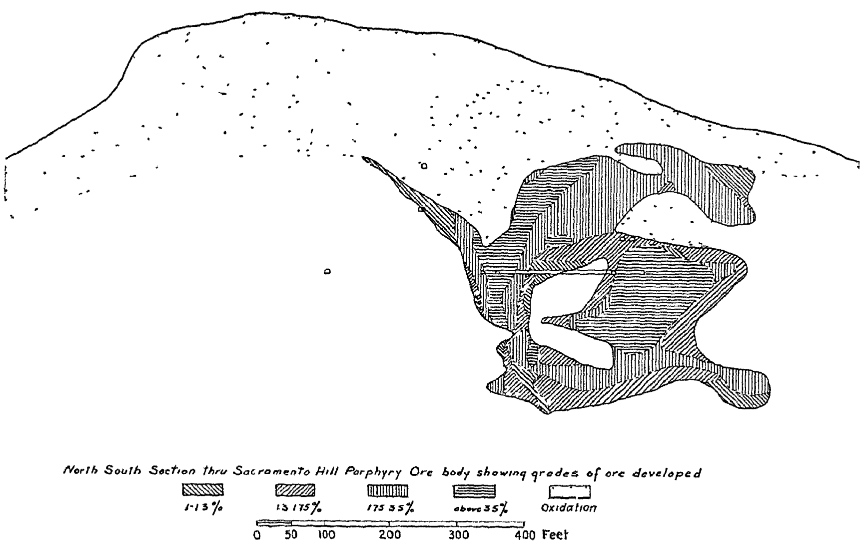


Plate 11.—Sacramento Hill Orebody.

Four convenient subdivisions can be made, dependent on the formation replaced, as follows: ores in porphyry, in contact breccia, in contact metamorphic limestone, and in relatively unaltered limestone. In con-

sidering these ores, the primary mineralization will be taken up first and the effects of meteoric water circulation left for later discussion.

Ores in Porphyry

To this type belongs the ore developed recently in Sacramento Hill. The ore occurs as a very irregular mass of chalcocite, pyrite and some bornite in brecciated, altered porphyry at the inner side of the contact breccia. The ore also penetrates the contact breccia, as shown in Plates 5 and 11, and connects with some typical bodies in this latter rock. The porphyry, in general, is sericitized, and somewhat silicified close to the

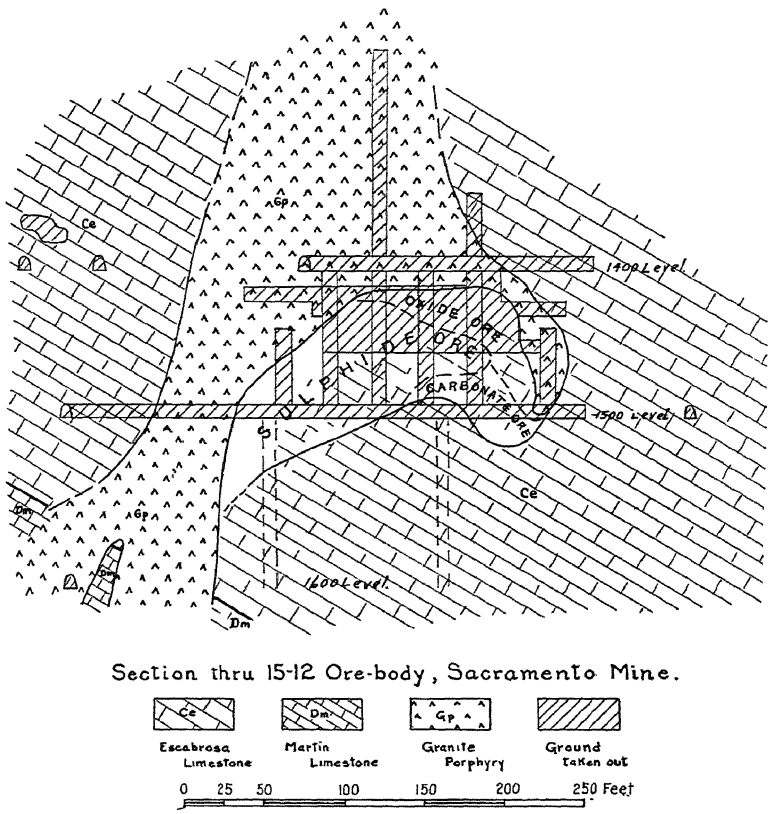


PLATE 12.—SHOWING CONTACT OREBODY ALONG SACRAMENTO DIKE AND SPECIAL FEATURES OF OXIDATION.

breccia, and in parts of the ore. The primary sulphide minerals are pyrite and bornite alone, this being the only type of ore in the camp in which chalcopyrite is not an important constituent. Partial analyses of this porphyry ore are given in Table 2.

Besides the ore in Sacramento Hill, ore occurs in porphyry in a

great many places in the contact-metamorphic zone, where the sulphides replace limestone and in a minor, but very distinct way, the adjacent porphyry. A noteworthy example of this is the orebody known as the Dividend Slice No. 1, where most of the porphyry is entirely converted into ore.

Ores in Contact Breccia

This second type of ore is found from the Holbrook to the Lowell, Oliver and Sacramento mines. The bodies are extremely irregular in shape and size, and are not confined to any definite horizon, but are usually found in unenriched and unoxidized state. One of these bodies outcropped to the west of Sacramento Hill.

The breccia, as has been explained, is highly silicified, and the ores themselves may be in an extra siliceous portion, or in a chloritized zone due to one of the later dikes mentioned before. The value of the contact breccia as an ore zone has been realized only within the last few years, as the rock is perhaps the most difficult of all to prospect.

Ores in Contact-Metamorphic Zone

To this class and the next belong a great majority of the ores heretofore mined in the district. The contact-metamorphic zone, marked by many alteration products in the limestone, but mainly by sericitization of both intrusive and intruded rocks, forms a block around Sacramento Hill and the associated extensions of the porphyry. The ores occur as irregular bodies intimately connected with dikes and sills which are direct offshoots of the main porphyry masses. The replacements extend from some distance in the porphyry, through some of the unrecognizable altered limestones, into the distinct bedding planes of less metamorphosed sediments.

Faulting and crushing previous to the introduction of the porphyry and ore solutions played an important part in determining the location and limiting the size of individual orebodies. Very little post-ore deposition faulting has occurred so little, in fact, that it is a negligible factor for most of the district.

The contact-metamorphic zone occurs almost entirely in Martin and in the first 200 ft. of Escabrosa limestone. Some orebodies, however, are found in the top 100 ft. of Abrigo, as shown clearly in Plates 5, 6 and 7. To this zone belongs the ore found along the Dividend Fault, which has played such an important part in structure and mineralization of the district. The rich ore mined early from the Czar and Holbrook was taken out along this fault, and rich ore is still being mined in the Czar from along it.

To this type of orebody belong also a great part of the ores immediately adjacent to the Sacramento dike, Lowell sill, and to dikes in the

Junction and Oliver mines, though these bodies, being outside of the highly altered zone, generally grade into typical replacements of the unaltered limestones.

Ores in Relatively Unaltered Limestone

To this subdivision belong the orebodies in the Southwest, Higgins, Wolverine, Uncle Sam, Shattuck, the south end of the Spray, Irish Mag and Oliver mines, all of the ore in the Cole and Briggs, and the major portion of the ores in the Sacramento, Hoatson and Junction mines. A brief description of each of these occurrences is given below.

The Southwest orebodies, which include the original ore discovery of the camp, occur as replacements of Martin and Escabrosa limestones along the Czar Fault. The original Queen orebody outcropped at the surface, at the big glory hole at the base of Queen Hill. It occurred in the foot-wall side of the Czar Fault, replacing Martin and Escabrosa limestones. Following the Czar Fault to the southwest is a continuous chain of orebodies in the hanging-wall side of the fault, in the same stratigraphic horizon, until the present Southwest workings are reached. At this point the orebodies again cut across to the foot-wall side, but keep in the same horizon. It is these foot-wall orebodies that are being mined at present. The Czar Fault is here broken up and forms blocks with east-to-west faults. In this crushed zone occurs a large mass of brecciated iron-stained silica, which outcrops on the surface on both sides of Hendrick's Gulch. The orebodies are found at and near the contacts of this mass, as replacements of Martin and Escabrosa beds, and penetrate the breccia itself for short distances. The ore is both of copper and of lead.

The ore early mined along the Czar Fault (and now being largely remined), was closely associated with porphyry dikes and sills, which came up along the fault. The present orebodies, however, are not in contact with porphyry. The silica breccia mass, however, in depth is connected with porphyry.

Following the Silver Bear Fault, which forms the eastern boundary of the Queen Hill Block, another series of orebodies occur, also as replacements of Martin and Escabrosa limestone. These orebodies are directly related to porphyry dikes and sills.

The Higgins, Wolverine, Shattuck and Uncle Sam orebodies occur close to and in contact with the Shattuck dike. The Higgins and Wolverine are the farthest west, and therefore farthest removed from Sacramento Hill. Here the dike, as exposed underground, occurs as a dike of unaltered porphyry, and the orebodies are found on both sides replacing Abrigo limestone about 100 ft. below the top. The Uncle Sam and Shattuck orebodies occur as replacements of Martin and Escabrosa limestones, very intimately associated with a large sill of porphyry with which the Shattuck dike connects in depth. At the Shattuck, another mass

of silica breccia occurs, more intimately associated with the porphyry than at the Southwest, with ore formed at the contacts, and penetrating into the mass. Here again the ore is both lead and copper as at the Southwest.

To the south and west of the Contact Block is a relatively up-thrown block, bounded by the Dixie-Howell-Senator Fault on the north and east, and by the Oliver Fault to the southwest, previously mentioned as the Oliver Block. In Irish Mag, Gardner, and Oliver ground, the block is much broken up by N.-S. fractures, and here there are orebodies in the foot-wall side of the Dixie-Howell-Senator Fault replacing Abrigo and Martin beds. In Spray, and in part of the Oliver ground where the block is less fractured, orebodies are found entirely as replacements of Abrigo limestone, from 100 to 300 ft. down from the top. In Spray ground no porphyry has yet been found anywhere near these orebodies. In Oliver ground, they are associated with small dikes.

In the Cole mine, orebodies occur in the Martin limestone and extend for a short distance into the top of the Abrigo, along N.-S. fractures. These orebodies occur in the hanging-wall side of the Oliver Fault. This fault, between the Cole and Oliver mines, has cut off one of these orebodies and is, therefore, post-ore deposition in age, one of the few breaks of its kind in the productive area of the camp. The Cole orebodies are not directly connected with porphyry.

The orebodies of the Briggs mine are similar to those of the Cole in structure, being related to a N.-S. fractured zone, not directly connected with porphyry. They are also replacements of Martin limestone.

The orebodies in the Hardscrabble claim of the Sacramento mine, and part of the Hoatson mine are directly associated with the Sacramento Dike. The orebodies exist as replacements of Escabrosa limestone, the horizon being about 200 ft. up from the base. The orebodies here are in direct contact with the dike on both sides, and make into it for short distances.

To summarize, the orebodies in the unaltered limestone occur for the most part at or near porphyry contacts, but they also occur away from porphyry in fractured country. In fractured country and where intrusions are numerous, the ore horizon tends to rise, and ore is found mostly occurring from the base of the Martin limestone to 300 ft. above the base of the Escabrosa. In unfractured, and relatively less intruded country, on the other hand, the horizon tends to drop and ore is found in the Abrigo limestone.

In the illustrations showing the zones of alteration (Plates 5*A*, 5*B*, 6*A*, 6*B*, 7*A*, and 7*B*) the orebodies in porphyry are seen to be between the sericitized and silicified portions of that formation. The ores in breccia are in a highly silicified zone. The ores in the contact-metamorphic limestones occur where the alteration is mainly sericitic.

Table 2 gives partial analyses of the orebodies along sections shown

by the illustrations and the rocks that they are found in, as well as the character of the ore dependent on oxidation.

TABLE 2.—*Partial Analyses of Orebodies along Sections of Plate 4*

Rock Replaced	Location	Oxidation Features	Cu, Per Cent	Fe, Per Cent.	S, Per Cent.	SiO_2, Per Cent.	Al_2O_3 Per Cent.	CaO, Per Cent.
		Section A-A						
Porphyry	East end Sac. Hill	Enriched	2.76	9.3	8.0	59.6	13.6	1 2
Porphyry	West end Sac. Hill	Enriched	1.8	9.4	9.8	60.5	11.4	0.9
Porphyry	West end Sac. Hill	Enriched	3.7	10.8	11.8	58.1	10.4	0 8
Porphyry	West end Sac Hill	Enriched	6.6	13.6	15 8	50.1	7.8	0.7
Porphyry	West end Sac. Hill	Enriched	7 8	17.0	20.1	44.3	6.0	0 7
Porphyry	West end Sac. Hill	Enriched	12.2	16.9	21.3	43.7	3.8	0 4
Contact breccia	Breccia West of Sac. Hill.	Primary	6 4	19.1	22.4	40.7	5.3	0 7
Contact breccia	Breccia West of Sac. Hill.	Slightly enriched	6.1	14.7	16 4	46.1	9.1	0 5
Contact metamorphic limestone	Holbrook	Enriched	8.8	22.7	?	21.8	15.1	0.9
Contact metamorphic limestone	Holbrook	Enriched	4.4	24.4	25.3	13.0	15.1	1.0
Contact metamorphic limestone (slightly altered)	Holbrook	Slightly enriched	5.0	31 1	35.2	15.2	6.2	
Escabrosa limestone	Holbrook	Residual carbonate	6 3	22.3	0.2	10.1	5.3	14.8
Devonian limestone	Southwest	Residual carbonate	4 1	12.2	0.1	18.3	7.3	12.9
		Section B-B						
Escabrosa lime and porphyry	Uncle Sam Mine	Residual ore	14.7	26.3	...	20.6	7.5	2 6
Devonian lime	Uncle Sam Mine	Residual ore	6 5	22.5	...	19.8	9.5	7.0
Abrigo limestone	Gardner	Primary	6.1	28.5	29.3	20.7	4.3	1.7
Devonian limestone	Gardner	Primary	4.7	32.5	36.0	16.1	6.1	3.1
Escabrosa limestone	Gardner	Residual carbonate	8 6	25.9	...	20.0	5.1	3.0
Metamorphosed Devonian	Gardner	Primary	4.3	34.1	27.7	14.8	2.45	1.3
Contact breccia	Gardner	Primary	4.8	30.6	40 0	13.6	4.5	1.4
Metamorphic limestone	Sacramento	Enriched ore	9.3	22.7	23.0	25.6	11.4	1.3
		Section C-C						
Abrigo limestone	White Tail Deer	Residual carbonate	5.25	13.8	...	45.9	5.2	2.6
Partly altered Devonian	Holbrook	Residual carbonate	8:1	13.2	Tr.	26.0	10.5	1.6
Partly altered Escabrosa	Holbrook	Residual carbonate	4.4	25.1	Tr.	19.4	9.5	1.5
Sericitized metamorphic	Holbrook	Enriched sulphide	5.5	21.1	?	14.9	23.0	0.4
Sericitized metamorphic lime and porphyry	Holbrook (Dividend).	Enriched sulphide	6.6	17.5	12.6	17.8	23 2	0.3

MINERALOGY OF THE ORES

As in the case of rock alteration, primary minerals in the ores have been changed greatly by the circulation of meteoric waters, and the criteria have already been given for determining the primary zone under

the caption, "The Effect of Contact and Hydrothermal Metamorphism on the Formations."

The original sulphides of copper deposited are comparatively few in number, chalcopyrite and bornite being the important ones economically. Tennantite is the only other copper-bearing mineral so far found in the primary zone, if exception be made of one or two unknown minerals seen with the microscope. Pyrite is such an important constituent of the ores, that it must be considered along with the copper minerals, and magnetite, hematite, sphalerite and galena are common enough to be considered part of the ores, especially the last two, since they are the source of oxidized ores of importance.

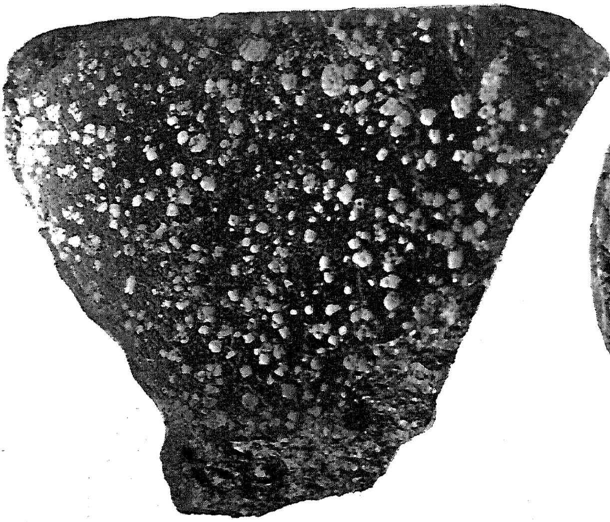
Chalcocite is very abundant in many forms in the zone of enrichment but has not been seen in the undoubted primary zone, though this has been considerably explored. It is therefore highly probable that this mineral was not formed by the ascending solutions in this district, and conflicting microscopic evidence in the ores above has been considered insufficient to offset the field evidence of the primary zone.

Pyrite

Of the sulphides, pyrite is generally the first formed, and accompanies every one of the stages of alteration described under metamorphism. In the ores, very fine examples of the structures described by Graton and Murdoch[8] may be seen, grading from the perfect "porphyritic" relation between well-formed pyrite crystals and the other sulphides, to a mass of pyrite grains with little interstitial chalcopyrite or bornite.

As additional evidence for explaining the mode of formation of some of the sulphides the following observations will be given. Around any orebody, pyrite usually persists in the same general relations to the rock or gangue, that it has to the other sulphides in the ore, with the exception that in the ore the individual pyrite grains are broken up, penetrated or reabsorbed by the other sulphides in such forms as are illustrated in Plate 13, Fig. 2, and Plate 14. Thus, ore in porphyry has pyrite veinlets extending into the rock along joints and it will be found disseminated through the rock in irregular grains that usually started around a biotite. These same pyrite veinlets and disseminated grains will persist even when all the rock has been replaced by sulphides, as can be seen in Plate 13, Fig. 3. In limestone replacements, pyrite will extend out along bedding planes in layers that are continuous with pyrite bands in the copper ore. And pyrite crystals will be found to have the same "porphyritic" relation to slightly altered Abrigo limestone as they have to the rich ore in

[8] L. C. Graton and J. Murdoch: The Sulphide Ores of Copper, *Transactions*, vol. 45, pp. 26–93 (1913).



FIGS. 1 AND 2.—"PORPHYRITIC" STRUCTURE OF ORES IN ABRIGO LIMESTONE (LEFT) AND IN BRECCIA.

FIG. 3.—REPLACEMENT OF ALL ROCK BY SULPHIDES WITH PYRITE VEINLETS PERSISTING.

PLATE 13.

that formation, with the exception given above, that there is little breaking up of the pyrite (see Plate 13, Fig. 1).

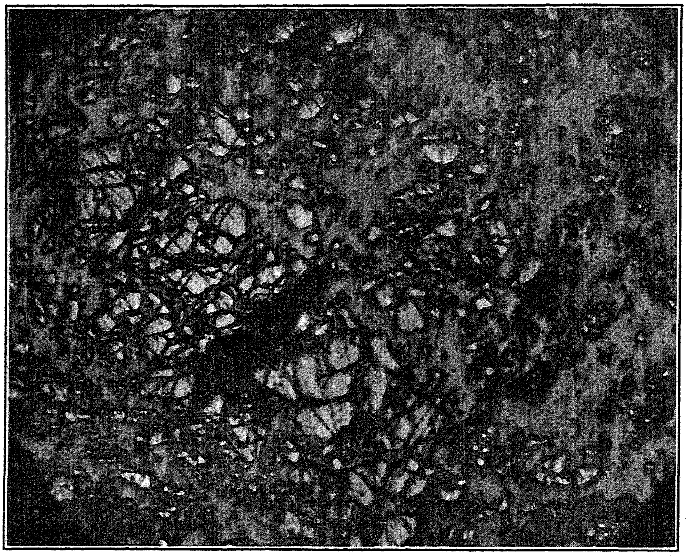


FIG. 1.

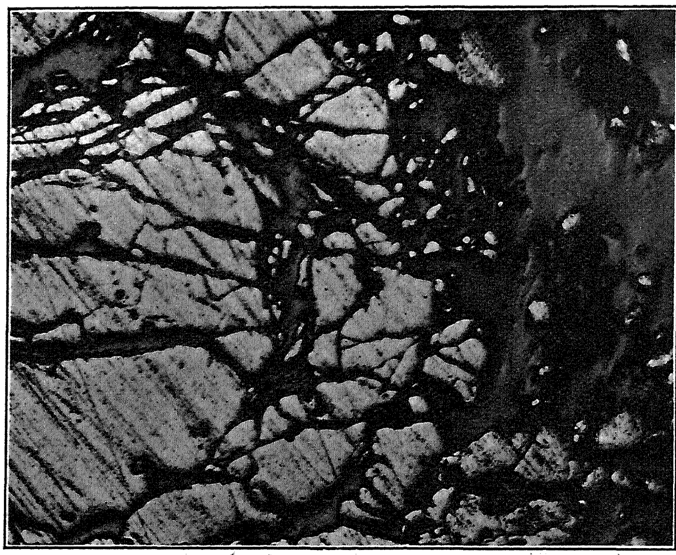


FIG. 2.

PLATE 14.—ORIGINAL PYRITE GRAINS BROKEN AND REABSORBED BY OTHER PRIMARY SULPHIDES.

This information, coupled with microscopic observations concerning the age of crystallization of the pyrite has led to the belief that in the ore

formation pyrite is the advance sulphide, formed partly by solutions that are not yet precipitating copper minerals. When these latter are formed, they take their place by replacement of the gangue and rock mostly, but also by slightly replacing the earlier formed pyrite. This may explain certain veinlet-like stringers of pyrite, which apparently cut the other sulphides, but which when examined in detail show that each grain of pyrite has been attacked by chalcopyrite or bornite.

Age Relationship of Pyrite.—In the porphyry ores, pyrite is found to have crystallized at about the same time as the sericitization occurred with some of the grains a little earlier. Pyrite deposition also extended into the period of silicification, but was undoubtedly also replaced by quartz. Bornite, the copper mineral, is found replacing the pyrite as already mentioned.

In the contact breccia, due to the many complicated interrelations found, the pyrite cannot be said to have formed at any definite stage. In general it holds true that pyrite is earlier than the rest of the sulphides, but distinct veins of it have been found through crushed pyrite, bornite and chalcopyrite ore. Magnetite, specularite, and sphalerite are of about the same age as the pyrite, and garnet, diopside and wollastonite slightly earlier.

In the contact-metamorphic limestone orebodies, pyrite and sericite again belong to the same period of formation, earlier than the copper sulphides, and slightly later than tremolite and other contact silicates. In this zone, pyrite veins are found cutting magnetite, specularite and earlier pyrite bodies.

In the orebodies that replace unaltered limestone the relations are the simplest, pyrite being closer in age to the copper, zinc, and lead sulphides than in any other place.

Chalcopyrite

This mineral is undoubtedly the most abundant primary copper sulphide in the district, and is prevalent in all the ores, except in the disseminated porphyry ore in Sacramento Hill. This ore is distinguished mineralogically from all others in the district by having as undoubted primary sulphides, pyrite and bornite alone.

Chalcopyrite is generally associated with the more siliceous ores, except in porphyry, its period of formation being slightly later than that of sericitization, but contemporaneous with silicification. In cases where there is late chloritization, some remarkably pure bodies of chalcopyrite are formed, as is also the case in certain vein-like bodies extending far above the general ore horizon, where galena is also abundant.

The intergrowths of chalcopyrite and bornite show them to be of contemporaneous precipitation, with a slight lagging of the bornite formation.

Bornite

This mineral is universally found in the primary ores, though not as abundantly as chalcopyrite, as has been stated.

Although bornite occurs undoubtedly in the richer primary ores, there has been no reason found for its formation in preference to chalcopyrite, as it comes in rich and lean ores, and in those with a great amount of pyrite, also when there is scarcely any pyrite. Fig. 1, in Plate 18, illustrates a case where bornite and pyrite replace the cement in a calcareous Devonian sandstone, where the amount of copper is not enough to give 0.2 per cent.

Tennantite

This mineral is found usually in microscopic amounts only, in practically all the ores. Its age of crystallization is between that of pyrite and chalcopyrite. There may be some tetrahedrite, but as arsenic is more abundant than antimony in the specimens analysed, the mineral seen has been taken to be tennantite.

Sphalerite

The sulphide of zinc is found in all the primary ores in minor quantities, with zonal inclusions of chalcopyrite and galena in very fine grains (Plate 17, Fig. 1). In a few places such as between the Southwest and Czar mines, in the Gardner, and in Briggs ground, sphalerite is found in rather large bodies with pyrite and chalcopyrite. It generally replaces Martin limestone, and apparently its general relations are no different from those of the copper ores.

Galena

Like sphalerite, galena is associated with most of the primary ores, but is decidedly more abundant where the ores are in highly chloritized ground. The ground adjacent to the latest basic porphyry dikes, in contact breccia and in limestone, generally has more galena than the rest. An exception to this may have been the zones around the silica breccia masses of the Southwest and Shattuck mines, but unfortunately oxidation has obliterated thoroughly all primary mineralization. Lead carbonates and sulphates are here found in abundance.

Galena is found more abundantly also in the top zone of the ores, as a residual sulphide, so the general impression given by the occurrence of the lead sulphide is that it is more abundant in the last stages of mineralization and also that it was carried farther than the copper.

Other Primary Metallic Minerals

Magnetite and specularite occur sometimes in large bodies that show a very perfect intergrowth of these oxides, as well as with pyrite, as has been mentioned. The occurrence of these bodies is limited to the contact-metamorphic zone, or is immediately on the contact of porphyry dikes. A few grains of pyrrhotite have been seen here, indicating the high temperature of their formation.

Specularite is also found associated with the orebodies in contact breccia that come with the basic, chloritizing porphyry dikes. The oxide is here a valuable indication of the proximity of ore, in a zone where all other features are baffling in their irregularity.

Silver and gold minerals have not been seen in the primary zone, with the possible exception of a few grains of polybasite found in galena. Gold and silver values are considerable in all the copper ores, and more so probably in the sulphides than in the oxides, but their amount could easily escape detection even under the microscope. Tennantite, as well as one or two brownish unknown minerals in the sulphides may be the precious-metal carriers.

IX. OXIDATION AND ENRICHMENT

General Features

The most striking feature of oxidation of the orebodies in the Warren district is the varying elevations at which the column, from oxidized to primary ores, repeats itself in going from the western to the eastern side of the mining area. At the extreme western end, the Higgins and Wolverine mines have primary ores in Abrigo limestone far above the level of the adjacent creek bottoms, and consequently far above the present ground-water level. In the Czar mine oxidation and enrichment go down to just under the 300 level, which is about 5,000 ft. above sea level. In the Holbrook division some primary ore outcropped at the west of Sacramento Hill, while some distance south, enrichment goes down below the 500 level. In the Gardner, primary ores are not found till the 9th level is reached, and in the the Lowell mine they are encountered, in some instances at the 10th level, but generally below the 13th, although the permanent water level was encountered here around the 10th level, or at an elevation of 4,325 ft.

In the Hoatson mine, oxidation features were found down to an elevation of about 3,800 ft. above sea level, and even deeper in part of the Sacramento and Junction mines, although the permanent water level was far above this before the mines were drained by opening up and pumping from the deeper levels of the Junction.

The oxidation features shown by the illustrations of the sections through the camp (Plates 5*b*, 6*b*, and 7*b*), illustrate this general inclination to the southeast. It will be noticed that the oxidation follows in general the dip of the Paleozoic sediments, the average horizon reached being somewhere in the Devonian. There is a marked lagging of the oxidation at the eastern end, however, as here it does not reach the top Devonian. In this, the oxidation may be seen to correspond more closely to the pre-Cretaceous erosion surface.

It will be seen also that there are some sudden changes in the level of oxidation, as best shown in section A-A of Plate 5, and also that there are great variations in the extent of the zone of secondary sulphides. All these features have fairly definite relations to the geologic history and structure, to the nature of the formations, and to the primary alterations in them. The object of this chapter will be to explain what is known of those relations.

It will first be necessary to present the criteria for the subdivisions of the zone in which meteoric waters have circulated, in many directions, but generally downward. It has been stated before that the primary zone was marked by steady conditions and constant mineralogical associations. The deepest effects of the surface-water circulation are immediately and clearly marked in this district by veinlets, filled cavities and coatings of other minerals.

Just above the primary zone there is an extremely variable thickness where the prevailing secondary mineral is siderite with a little gypsum and an extremely small amount of chalcocite. Above this is usually a zone where halloysite and kaolin are most prominent among the secondary gangue minerals, and chalcocite is abundant as a replacement of primary sulphides. Next higher, limonite, and the staining due to iron oxides—gossan formation, in other words—is the prevailing feature. Within this zone there are found the local bodies of residual ores, which have for some reason resisted the general leaching effect of strongly oxidizing waters. One or all the zones above the primary may be lacking at any point for reasons which will be explained later.

Relations of Oxidation to Structure and Geological History

Notman[9] has already pointed out the relations between the attitude of the oxidation and present and past water levels. That the present water level does not account for the oxidation is proved by the occurrence of unoxidized ores above that level in formations that are in other places completely oxidized also above that water level. This is illustrated by the primary ores in Abrigo at the Higgins mine, and the completely oxidized Abrigo ores in the Wolverine and White Tail Deer mines.

[9] *Transactions of the Institution of Mining and Metallurgy*, vol. 22, p. 561 (1913).

Further proof is had in the presence of gossans several hundred feet below the water level in the Lowell and Hoatson mines.

The general plane of the oxidation corresponds very closely to the known pre-Cretaceous erosion surface, and the tilt of the oxidation is, therefore, the present tilt of the Cretaceous sediments. If the ground-water level at the time of oxidation had been an inclined plane that followed the Devonian horizon, due to the permeability of the purer limestone above, there would not be abundant unoxidized orebodies in Escabrosa limestone in the Sacramento and Junction mines. And if the ground-water level had stood several hundred feet lower in those mines at some post-Cretaceous time and had risen, say, due to valley fillings, it is probable that the ground water, now directly over that rise, would stand horizontal, and would keep to the old plane, or even a lower one farther west, which is far from being the case.

There is field evidence of another form to show that the oxidation was pre-Cretaceous, and this is the presence of gossanized fragments of porphyry, as well as fresher ones at the base of the Glance Conglomerate. The exposed pre-Glance topography also shows hard gossan reefs around the southeast side of Sacramento Hill.

Additional weight is given to the idea of pre-Cretaceous oxidation by the geologic history of this region. A very long erosion and a consequently long oxidation period is recorded between Pennsylvanian and Cretaceous times by the disappearance of the Paleozoic column off of the north side of the Dividend Fault. And while the granite porphyry intrusion may not have taken place till a good part was gone, the nature of the rock as well as of the later minerals formed presuppose a considerable covering of sediments which were removed by erosion before Cretaceous time.

It is, therefore, probable that the main part of oxidation and enrichment as found at present took place when the Paleozoic sediments were almost horizontal, the only tilt noticeable being slightly to the southeast.

There is very little evidence of superimposed oxidation since Cretaceous times, due very probably to the later ground-water level being higher than the first. Narrow, limonitic seams, and very slight enrichment on primary sulphides in the west end of the camp are the only visible effects of this oxidation. The post-ore faulting which is also post-oxidation in most cases, is not found very frequently in the thoroughly explored portion of the district, but will very probably be found more frequently when operations extend into other blocks.

Relations of Oxidation to the Formations and to Metamorphism

The circulation of meteoric waters has been very much influenced by the material they traversed, and this was dependent upon the formation and the degree of alteration.

In the Carboniferous limestones the massive purer beds had a tendency to crack and break rather than to fold, and waters could dissolve out channels through which extensive oxidation could be carried down rapidly. The same applies to the purer lime beds in the Devonian and Cambrian formations.

The more impure beds, which are also more thinly laminated, had a tendency to crush and fold, and while as a whole they may be found more altered by meteoric waters, it is because they retarded and distributed the effects all through the rock. This is noticeable in the Martin and Abrigo formations especially, the Abrigo being by far the most impervious.

The ability of meteoric waters to circulate and oxidize quickly in the Escabrosa limestone has made possible the abundance of residual ores in this formation. These are mostly carbonates fixed in this form against total leaching by the continually circulating cold waters. In a great many cases, however, the residual ore is found in the form of sulphide, mostly chalcocite, and also, in rare instances, as the original chalcopyrite and bornite. These sulphides are called residual because they are far above the level of normal enriched sulphides, and they have limonitic gossans or carbonate and oxide ores under them. Usually they are very pure, rich sulphides of copper, and are in the primary zone of sulphide replacement where the adjacent rock is pure calcite and contains little or no pyrite. Occurrences like this are found in the limestones all the way to surface where the main ore zone may be a thousand feet below.

The same quick circulation through the Carboniferous limestones has usually brought the zone of complete oxidation directly against the primary ores, with a thin intervening zone of secondary sulphides, which, as will be shown later, is not one of enrichment, but of leaching of values. The slower circulation in the impurer Devonian and Cambrian produced fewer residual orebodies, but a deeper zone of enrichment.

In the porphyry and contact breccia the differences in the effect of secondary alteration are far more noticeable, as can be seen from Plates 5*b* and 6*b*. Both the porphyry and breccia retard the meteoric water circulation, the breccia almost entirely, and the porphyry making it slow enough so that sulphide enrichment is at its most efficient point.

A special case which illustrates the difference in the penetration of oxidation through different rocks is given in Plate 12, which is an east-to-west section through an orebody on the 1,500 level of the Sacramento mine, at present being extracted. This ore occurs along the Sacramento dike, here cutting Escabrosa limestone and having a projection downward to the east as shown. The ore replaces the limestone, which is not appreciably metamorphosed at a distance from the porphyry. The replacement is very perfect along the beds, as certain cherty bands are left untouched in the midst of the ore.

The oxidation feature that is noticeable is that the ore is almost unaltered close to the main dike, enriched farther away, converted into mixed oxide and sulphide under the thin portion of the porphyry, and entirely into carbonate next to the limestone. The water circulation here must have come partly from the east and partly from underneath through the limestone, as the porphyry ordinarily presented an effective barrier to circulation perpendicular to the beds before the tilting.

The oxidation features are even more noticeably related to the processes of primary ore alteration than they are to that of the formations.

Intense silicification such as is found in the contact breccia and in a great part of the contact metamorphic limestone of the Gardner mine made circulation so slow that in these places the primary ore zone remains at a level far above those of adjacent occurrences. Also, the oxidation is so complete where there is any meteoric circulation at all that the zone of leaching comes to within a few feet of the primary zone, with only a thin layer between where covellite is most abundantly found in the district. These relations are well shown in Plate 5, Section A–A just west of Sacramento Hill.

Sericitization produced rocks in which the circulation was slow, but in which the secondary waters had opportunity to remain acid, as nearly all the lime was removed in the primary alteration and pyrite was abundant all through the rocks. Under these conditions the depth of the zone of secondary enriched sulphides was greatest, and the enrichment most efficient. At the same time, in the rocks very much sericitized there are no residual carbonates found, due, of course, to the lack of lime. If, however, the ore formation extended into unaltered limestones above those sericitized, carbonates might be found higher, as is the case in the Uncle Sam mine. Native copper and oxides are developed to a great extent in a shallow zone between the gossans and enriched sulphides in aluminous ground due to primary sericitization.

The ores in Sacramento Hill porphyry, and in the zone of contact-metamorphic rocks around the main intrusives are the best examples of the coincidence of sericitization and deep sulphide enrichment, as can be seen in Plates 5*a*, 5*b*, 6*a* and 6*b*. The amount of enrichment even in these cases has not been exceedingly great, as the original ores were rich to begin with, and very pyritic bodies have remained poor, first because pyrite is replaced with so much more difficulty by chalcocite than bornite and chalcopyrite, and second, because pyritic masses are generally quite solid, more siliceous and impervious.

The ore developed in Sacramento Hill is extremely irregular in shape and grade and owes its irregularity more to primary mineralization than to secondary enrichment, because so far as has been seen, the relative amount of primary bornite left in parts at the top of the ore close to the oxidation line is about the same as at the deepest levels. A section show-

ing the ore grades is given in Plate 11, worked out from all information available. This body has not yet been mined extensively. The grades of ore can be seen to bear very little relation to the limonitic oxidation.

The orebodies classed as sulphide replacements of unaltered limestones are ordinarily found in their primary state or as residual carbonates. Oxidation is rapid through unaltered rocks, and waters usually carry enough lime to fix the copper as carbonate. In such a zone, carbonates are apt to be found to the very top of the mineralized zone, many times above hundreds of feet of gossanized ground.

TABLE 3.—*Showing Efficiency of Enrichment by Descending Waters*

Orebody	State of Oxidation	Tons per Cu. Ft.	Pounds Cu per Ton	Pounds Cu per Cu. Ft.
	Sericitized Contact Metamorphic Zone			
14–7. ..	Primary. ...	0.1143	126 2	14.42
13–13 . . .	Enriched sulphide .	0.1008	189.8	19.10
	Sulphide Replacement Orebodies in Abrigo			
12–3–18..... ...	Primary...... ...	0.117	104.0	12 17
13–10....... ..	Enriched.	0.0984	147.0	14.46
White-Tail Deer. .	Residual carbonate....	0.0578	142 0	8 20
	Sulphide Replacement Orebodies in Martin and Escabrosa			
9–8–58.	Primary	0.1255	130.0	16.31
8–17–11..... ...	Enriched and oxide. .	0.098	112 0	10.97
8–17–4......... .	Residual.............	0.0609	113.0	6.88
8–16–5, 8–16–16.. .	Residual	0.0609	137.0	8.34
8–16–18 .. .	Residual..	0.0609	108.0	6.58
	Sulphide Replacement and Contact Bodies in Escabrosa			
1300 Junction	Primary........ ...	0.1111	116.0	12.90
16–3–39..	Primary..	0.140	140 2	19.63
16–3–68	Partly enriched .. .	0.1267	192.4	24.38
15–12 Sulphide.....	Secondary sulphide.	0.115	130.0	14.90
15–12 Oxide.......	Oxide.....	0.08	224.0	17.92
16–3 Oxide.......	Oxide............ .	0.08	268.0	21.40
15–6–1, 15–6–9. ...	Residual carbonates..	0.065	170.0	11.05
15–13....	Residual carbonates..	0.065	188.0	12.20
14–20.............	Residual carbonates...	0.065	114.0	7.41

Table 3 has been prepared to show approximately the efficiency (or lack of it) of enrichment due to descending meteoric waters.

Orebodies have been taken for analysis which were originally of similar composition, as far as knowledge of them goes, and from mining records kept for a period of several years, the contents of copper in pounds per cubic foot in place has been figured. This is of course a figure for the ore mined, and must differ somewhat from the actual content of all the ground in an orebody, depending on the method of extraction used. All errors have been considered, however, and the relative value of the figures is about correct. Unfortunately the ores in the breccia and silicified ground are all primary and no comparisons can be given, and all the ore in porphyry is enriched.

From these tables it will be seen that the copper content per unit of volume is increased materially in the sericitized contact zone and in the more aluminous Abrigo formation where the secondary sulphides are found. Chalcocite formation is accompanied by leaching out of values in the replacement bodies, and all the residual ores show dissipation of copper contents. The narrow oxide and native copper zone is generally the richest of all.

Gossans

In regard to the gossans formed in the different zones of primary alteration, they can be said to extend downward for the greatest distances in the more permeable ore zones—this is true for those classed as sulphide replacements. These gossans are likely to contain residual ores anywhere. The thoroughly oxidized portions, where the formations were silicified or sericitized or chloritized are apt to be more completely leached out, and penetrate the ore zone least in highly quartzose ground. In composition the gossans of course reflect their source.

The only generalization that can be made about gossans derived from lean sulphides and from pyritic copper ores is as follows. The mixed sulphides of copper and iron are more readily attacked by oxidizing solutions than the straight iron sulphides, therefore producing stronger solutions. These stronger solutions have been found to have a segregating action on the components of the surrounding rocks, separating to some extent silica, alumina, and lime, and also depositing the iron oxides in purer form. The gossans from pyrite alone have been observed to leave the gangues and oxides of iron in about the same mixtures as in the original primary material.

These observations are of course susceptible of an infinite number of variations depending on the original composition of sulphides, of their gangues, and of the surrounding rocks, but in general they can be used to some extent in judging the kind of ground under an oxidized portion of an ore zone.

Minerals of the Zones of Enrichment and Oxidation

Siderite

Under the zone of secondary enrichment the presence of siderite has already been noted. This mineral occurs sometimes in veinlets replacing either sulphides or chloritic minerals or calcite. At other times siderite is found in a curious box-work honeycomb structure as if it had replaced laminated rock along certain planes and the intervening material had been later dissolved out. The ferrous carbonate has a distinct selective action in replacing a rock, as it usually leaves intact pyrite grains, which then appear contemporaneous with the siderite. Sphalerite and galena are also found more abundantly with the siderite than around in the unaltered formation, and this fact has pointed to the possibility that these two sulphides may have been here deposited by secondary solutions.

Copper minerals in the siderite zone are very faintly coated with chalcocite. Gypsum is also found in vugs, even deeper than siderite, but apparently in late circulation channels.

Siderite probably formed as an advance mineral, as oxidation progressed downward. This may be proven by the closely similar forms taken by limonite in some gossans above enrichment, and the presence of siderite cores in that limonite. The iron carbonate has played an important economic role in serving as a precipitant for rich cuprite ores in the Czar mine, where copper-laden solutions have encountered a flat bed in Devonian limestone previously replaced by siderite and kaolin, and have precipitated cuprite and native copper in considerable amount. Throughout the district cuprite crystals can be found on oxidized or partially oxidized siderite.

Chalcocite

In the zone of sulphide enrichment chalcocite was the only secondary mineral of copper formed to any extent. It was usually formed by replacement of the bornite and chalcopyrite alone, leaving the pyrite untouched, or only slightly coated. The broken up, replaced pyrite grains in the primary ore remain in just about the same amount and with the same relations to chalcocite that they had to the primary minerals. When chalcocite and pyrite are the only remaining sulphides, it is difficult to believe that the iron mineral has not been abundantly attacked by secondary action. Pyrite has been replaced to a considerable extent, however, in the enriched ores of the contact-metamorphic type, by a sooty variety of chalcocite that possibly contains considerable covellite, since the latter has been seen coating pyrite grains in these ores.

In replacing the primary minerals chalcocite behaved in different ways depending on the general feature of the orebody. In the ore of

Sacramento Hill, pyrite is barely coated in the very sericitic portions of the porphyry, and is not affected appreciably in silicified rock. The selective action in replacing bornite is better marked here than anywhere

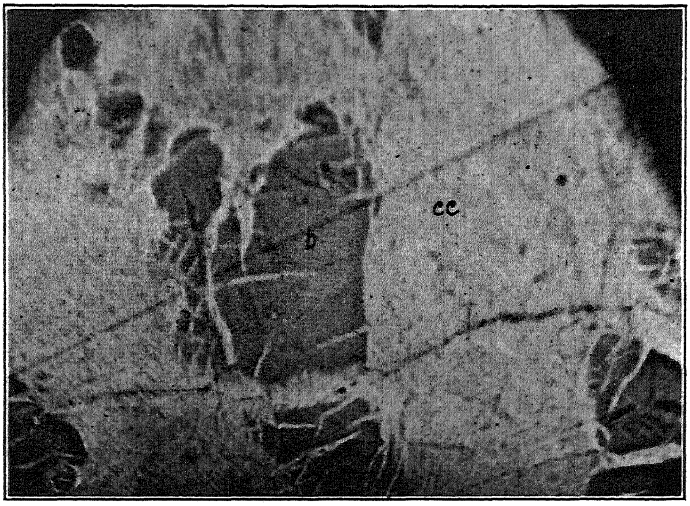


FIG. 1.

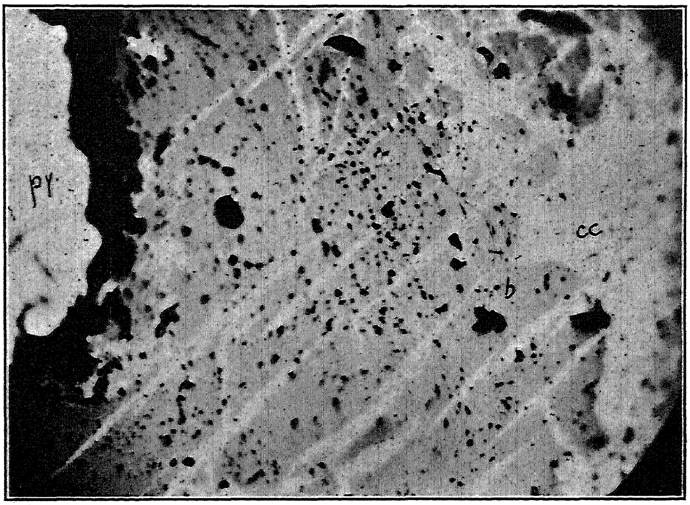


FIG. 2.

PLATE 15.—CHALCOCITE, *cc*, REPLACING BORNITE, *b*. PYRITE, *py*.

else as is also the replacement along crystallographic lines of the bornite in the arrangement commonly known as "lattice structure." This form of alteration grades into the usual one of veinlets and irregular remnants, as is shown in Plate 15. The field occurrence of this "lattice

structure" alteration has been quite carefully followed in the district and found everywhere to coincide with slow or incipient enrichment in rich ores, and where the selective enrichment of bornite in place of chalcopyrite is well marked. This has led to the conclusion that it is a delicate process in which crystallographic weakness can be of some influence. Etching of the chalcocite thus formed shows isometric figures, probably due to included or dissolved cupric sulphide,[10] but there are also stringers in which the chalcocite has orthorhombic cleavage, and these stringers can be seen to form a network of veins. The proximity and association of this orthorhombic chalcocite to major cracks and circulation channels, where even limonite is formed, points to its being due probably to the working over and recrystallization of the copper sulphides first derived from bornite.

Chalcocite of the "sooty" variety is found most commonly in the enriched ores of the zone of contact-metamorphic limestones. Massive chalcocite is here rare, and then only at the top of the zone of enrichment, with considerable native copper present.

In the ores that replace limestone, the formation of chalcocite is accompanied by leaching and formation of abundant cavities in the ore, a process that is reflected in the decrease of values as previously shown. This same thing is true in ores that have a very siliceous gangue, such as those in contact breccia. Pyrite is here oxidized to limonite even before the copper minerals are replaced by chalcocite, as is shown in Plate 16.

In the Southwest mine there is an occurrence of chalcocite that is somewhat different from the usual. The Devonian-Escabrosa contact beds are replaced by silica, almost chalcedonic in character, accompanied by abundant fine flaky specularite. This siliceous mass connects up with the main silica breccia replacement in this mine.

The top Devonian beds under those silicified have been hardly altered at all, except for a very slight amount of quartz addition, some recrystallization of calcite in stringers, and the formation of very fine veinlets as well as disseminated grains about a millimeter in diameter of what was apparently bornite and chalcopyrite, now replaced almost entirely by chalcocite. The original copper minerals have been found partially replaced by chalcocite in the center of coarse calcite veins that protected them. Pyrite is very rarely found.

The formation of chalcocite has taken place in this instance under conditions which would hardly allow the circulation of acid waters. A considerable amount of carbonates has formed, but in general the sulphide is relatively abundant. The possibility of chalcocite formation with the acid generated in the replaced mineral itself is indicated in this orebody, though this is by no means perfectly clear.

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[10] Posnjak, Allen and Merwin: *Economic Geology*, vol. 10, No. 6.

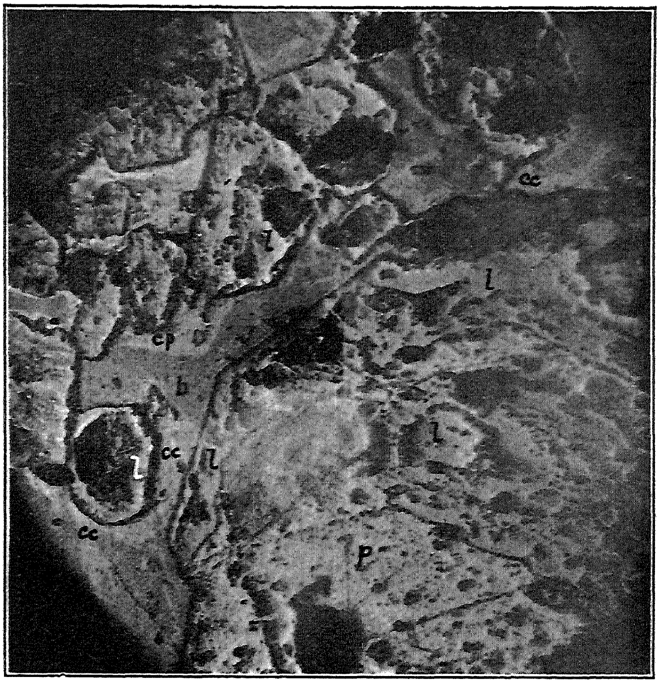

FIG. 1.

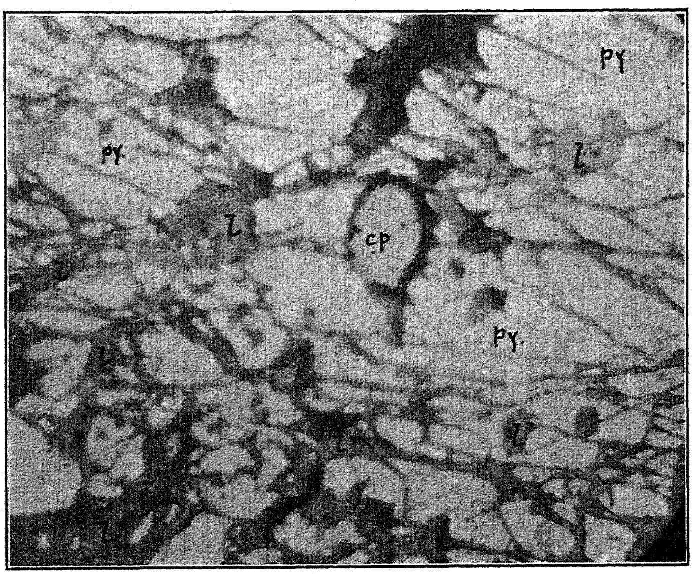

FIG. 2.

PLATE 16.—SHOWING OXIDATION OF PYRITE, *py*, TO LIMONITE, *l*, BEFORE REPLACEMENT OF COPPER MINERALS BY CHALCOCITE, *cc*.

Covellite

Covellite is formed in very minor amounts especially in the narrow oxidized zone above the contact breccia ores. It replaces chalcopyrite in preference to bornite, and is found always in microscopic needles. The only other occurrences of this mineral are where the enriched zone touches highly silicified, not very sericitic ores.

Bornite and chalcopyrite have been found in many ores of the district, formed by secondary agencies, as transition products, but they are of no economic importance.

Gangue Minerals

The gangue minerals that accompany the enriched sulphides are halloysite, kaolin, gibbsite, alunite, and serpentine with mixtures of quartz, calcite and the oxides of iron. Of the aluminous minerals, halloysite seems to be the most common, especially in the zone of highly altered limestones. Where the rock is sericitized so that the mica flakes are well formed, sericite persists unaltered through the process of enrichment and oxidation.

Oxide Ores and Native Copper

In the oxide ores just above the secondary sulphides, cuprite, native copper, tennorite, melanochalcite and delafossite are found, but the first two are the only ones of economic importance. The association of cuprite and siderite has already been given. The zone of oxide has in many instances encroached entirely on the sulphides, leaving only a remnant here and there of the latter. Flat orebodies along the limestone bedding are especially susceptible to this process.

The minerals of the residual orebodies are mostly the carbonates malachite and azurite, with some chrysocolla, aurichalcite and brochantite. Besides these, there are the sulphides already mentioned.

Carbonates

The association and the occurrence of the carbonates has been very well described by Dr. Douglas[11] and by Ransome[12] and no additional features can be given, except to repeat that these minerals are found all through the oxidized portion of the ores that replaced unaltered sediments, and are still a very important part of the ore reserves of the district.

Other Ores

In recent years oxidized lead ores have become more and more important, especially from around highly siliceous replacements in Martin and

[11] *Transactions*, vol. 29, pp. 511–546 (1900).
[12] *Loc. cit.*, p. 125.

Escabrosa limestone. Most of this ore has come from the Southwest mine and occurs at the edge of the siliceous breccia, as cerussite with some

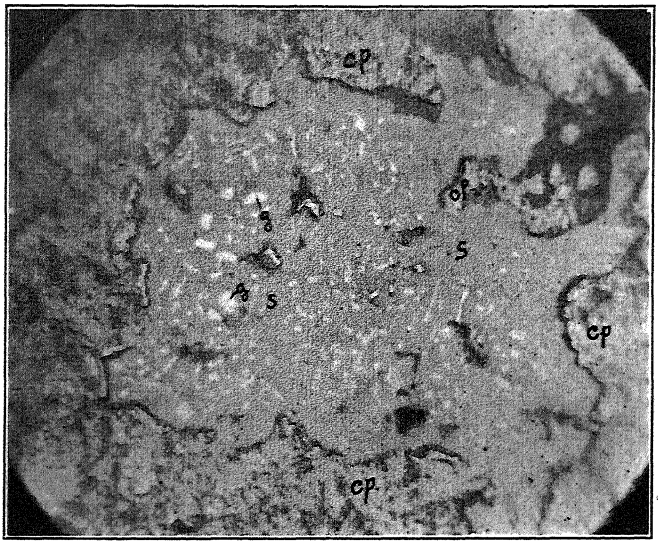

FIG. 1.—SPHALERITE, *s*, AND GALENA, *g*, INCLUDED IN GRAINS OF CHALCOPYRITE, *cp*.

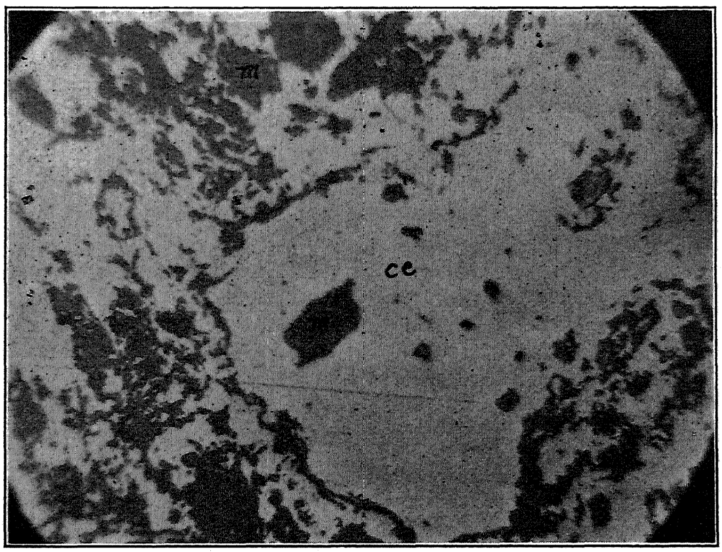

FIG. 2.—CHALCOCITE AND MALACHITE. ALTERATION RESEMBLING INTERGROWTH.
PLATE 17.

anglesite and variable silver and gold values. The primary minerals from which this ore was derived are unfortunately not present now, but

it is very probable that the main one was galena, which, as has been before stated, is apt to come in vein-like masses above the general level of the copper ores. Lead ores have also been found with similar associations in the Gardner, Lowell and Briggs mines.

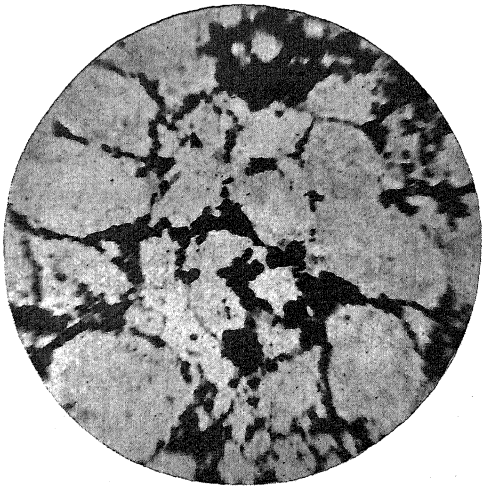

Fig. 1.—Bornite and Pyrite Replacing the Cement in a Calcareous Sandstone.

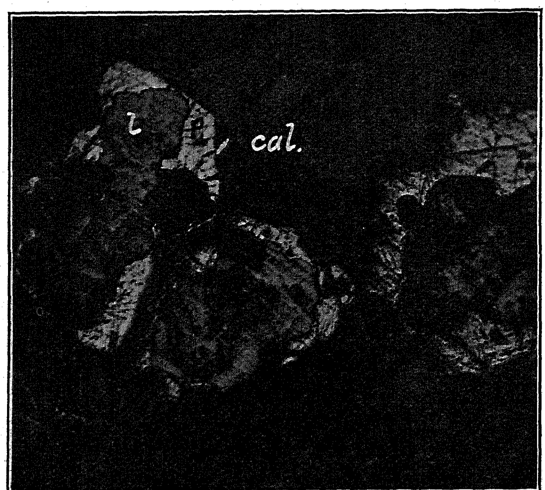

Fig. 2.—Pyrite Replaced by Limonite.
Plate 18.

Vanadium, in the form of cuprodescloizite has been found in the Shattuck mine in considerable amount, and rarely in the Dallas and Sacramento. The occurrence may prove in future to be of economic importance.

Manganese ore, in the form of psilomelane and braunite, with little pyrolusite, has been mined lately from several places on the surface of the district. It is found in irregular bunches as a replacement of limestone beds in the Naco or Escabrosa, close to the contact breccia, or to highly silicified outcrops of the sediments. The origin of this manganese is not yet known, for although manganese is found quite abundantly with some of the carbonate ores underground, it is here in the form of soft earthy pyrolusite due apparently to the concentrating action of oxidizing water.

Bibliography

The most important published works on the geology of the Warren district are the following:

James Douglas: The Copper Queen Mine. *Transactions*, vol. 29, pp. 511–546 (1900).
Frederick L. Ransome: Geology and Ore Deposits of the Bisbee Quadrangle. *U. S. Geological Survey Professional Paper* No. 21 (1904).
Bisbee Folio, U. S. Geological Survey, No. 112 (1904).
Arthur Notman: The Copper Queen Mines and Works, Part II. *Transactions of the Institution of Mining & Metallurgy*, vol. 22, pp. 550–562 (1913).
W. L. Tovote: Bisbee, a Geological Sketch. *Mining & Scientific Press*, vol. 102, pp. 203–208 (February, 1911).

Mention must also be made of the report on the mining geology of the Copper Queen property by John Mason Boutwell prepared in 1908–1909 for the use of the mines, one of the results of this work being the establishment of the present Geological Department.

Acknowledgments

To the Copper Queen Consolidated Mining Co. for its support and encouragement in the preparation of this report, the greatest thanks are due.

Arthur Notman, Chief Geologist of the company, has lent the utmost assistance, contributing from his store of knowledge gained by long and careful investigation in the district and in this part of the Southwest as a whole.

Former members of the Geological Department, and our colleagues of the Calumet & Arizona Mining Co., have also contributed materially to the information presented.

To Harvard University, and Prof. L. C. Graton in particular, we are indebted for being able to present the result of extensive petrographic and metallographic investigation, as 2,000 specimens from the Copper Queen geological collection were studied by one of the authors in the geological laboratories at Cambridge. Some of the photomicrographs were taken at the University of Arizona, for which thanks are especially due to Prof C. H. Clapp.

Assistance has been received from the engineering department of the company in the way of maps and stope records, and from the chemical department in the form of the analyses that are given in the report. Thanks are due to the operating department of the mines for a great deal of information, also for coöperation in pointing out and solving most of the geological problems of the district.

Discussion

Ira B. Joralemon, Warren, Ariz.—This paper has covered the situation so well that I have few suggestions to offer. Two points, however, are shown more clearly in the lower mines of the Calumet & Arizona Mining Co. than in Copper Queen workings, and may be worth mentioning. The first is the usual association of high-grade primary sulphide orebodies with much larger masses of pyrite and silica, averaging less than 1 per cent. copper. The richer areas occur sometimes around the borders of the lean pyrite bodies, and sometimes entirely within them. While as yet we have not been able to find any certain reason for the location of high-grade sulphides in low-grade masses, it seems likely that the age of fracturing is one controlling feature. The first stage of mineralization, with pyrite and silica, had the effect of sealing up the rock. The later solutions, richer in copper, could penetrate into the low-grade masses and bring in the rich copper sulphides only along fracture zones which were kept open by a more or less continuous motion during the time of mineralization. Fractures of the same age as the mineralization are therefore most favorable indicators of rich primary orebodies.

The second point I should like to emphasize is the very irregular pre-Cretaceous topography. This is best shown by drill holes east and south of the productive area. In the basin between Lowell and Warren, about 5,000 ft. east of the Briggs Shaft, drilling showed from 500 to nearly 1,000 ft. of Cretaceous conglomerate, filling an old canyon in the Carboniferous limestone. In the large valley south of Warren there was a far greater canyon. A drill hole about 4,000 ft. south of the Country Club, or 6,000 ft. south of the crest of the hills below Warren, was sunk to a depth of over 3,200 ft. without reaching the bottom of the Cretaceous conglomerate. This is the deepest diamond-drill hole ever put down in the Southwest. From the top of the Carboniferous limestone hills south of Warren to the bottom of this drill hole there is a difference in elevation of nearly 4,500 ft., in 6,000 ft. horizontally. Allowing for erosion of the hills and for considerable probable depth of conglomerate below the bottom of the drill hole, the total depth of the pre-Cretaceous canyon must have been 7,000 or 8,000 ft. This is pretty nearly the record canyon in history. With such a varied land surface, the pre-Cretaceous

oxidation may have reached a level far lower than the present deepest mine workings in the Warren district.

F. L. RANSOME, Washington, D. C.—There are a few points in the excellent paper just read upon which I should like to make brief comments.

The authors correlate definitely the thin quartzite at the top of the Abrigo at Bisbee with what I have named the Troy quartzite in the Globe-Ray region. I have myself suggested this correlation in a recent paper but think that more work is needed before it can be concluded that they are the same.

A very important and interesting feature brought out clearly in the paper just presented is the relation of oxidation and enrichment to the former Cretaceous surface. In my report on the Bisbee district published many years ago, I said that a good deal of the enrichment had probably taken place before the deposition of the Cretaceous Glance conglomerate. The underground workings, however, were not extensive enough at that time to show how regularly the zones of oxidation and enrichment dip south in conformity with the general dip of the basal Cretaceous beds.

The statement made in discussion by Mr. Joralemon regarding the great apparent thickness of Glance conglomerate found in drilling in the plain south of the Bisbee hills should be considered in connection with known overthrust faulting in that general region. The conglomerate may be both faulted and tilted, so that a drill section is not necessarily a measure of original thickness.

The observations made by the authors on contact metamorphism at Bisbee agree substantially with my own results. Although the Bisbee ore deposits are generally regarded as of contact-metamorphic origin they can hardly be considered as typical examples of that class. Such metamorphic silicates as are present are generally small or microscopic.

L. C. GRATON, Cambridge, Mass.—When the summer session of the Institute was held in Butte 3 years ago, we were given a paper by Mr. Sales on the geology of that remarkable district which may, I think, properly be regarded as constituting a new milestone in the advance of the application of geology to mining, in that it represented the accumulated data and evidence and speculation that had resulted from systematic, intensive and constant study by the geologic staff of the Anaconda company for a period of some 15 years. Obviously, such advantages for observation, interpretation and verification afforded to conclusions a validity that could have been secured in no other way. Without question, Mr. Sales' paper will continue to be the standard authority on the Butte deposits for a long time to come, and even when the horizons on which its observations were based may have become exhausted and abandoned, the principles which it disclosed will undoubtedly be found

in large part applicable to the deeper regions to which operations may extend.

The present paper, dealing with a district of such extensive development, such great production, such rich ores and such complex deposits as Bisbee, is certain to be of interest to all engaged in the study of mining geology. Like the paper on the Butte deposits, this one also is the product of systematic and steady effort for a number of years by the members of a company geologic staff, aided by the coöperation of their colleagues in the district.

Of the many directions in which this account of Bisbee geology merits attention, the feature that perhaps impresses one most strongly concerns the method, or at any rate, one of the methods, by which the results were achieved. Without in any way minimizing the difficulties of geological accomplishment at Butte, it may be said that, owing to the essential monotony of rock character and to the nature of the fractures in and along which the ores have been either deposited or dislocated, the outstanding problem in that district is of a structural kind. In Bisbee, the variety of rocks and the consequent variations and irregularities of the channelways that traverse them afford an ample supply of structural problems which the geologist must solve. Yet superimposed on these are problems of another sort and of great complexity; they are essentially chemical or mineralogical in character and relate not only to the initial deposition of the ore but to its subsequent alteration and to the enrichment or dispersal of its metal content. Problems of this kind have to be attacked by different methods from those employed for the solution of structural problems. The method which has been found most effective depends upon the microscope. No reader of the paper by Messrs. Bonillas, Tenney and Feuchère can fail to note the extent to which they have utilized the facts revealed by microscopic examination.

In this respect, therefore, it seems to me that this contribution marks still another step in the progress of scientific ore-finding. For although the method employed has been used by others, this, so far as I am aware, is the first recorded instance in which field methods and laboratory methods have been systematically coördinated on a comprehensive scale under the advantageous opportunities for constant and extended observation that are open to the geological department of a large mining company. The authors certainly deserve our congratulations and our thanks, and I think the Copper Queen company and other companies pursuing the same policy are to be commended for their far-sightedness in recognizing as valuable and permitting their geologists to follow methods of attack which some of narrower outlook might regard as too "flossy" and scientific to be of any practical service. Much is undoubtedly to be expected of such a practical, commercial application of this

combination of methods which has received so gratifying a start in the present paper.

In connection with one particular detail covered by the paper and referred to by Dr. Ransome—the subject of contact-metamorphic ore deposits—I should like to express assent to the statement of the authors that no sharp dividing line can be drawn in Bisbee between the type of deposit which nearly everyone would probably agree should be called a contact-metamorphic deposit and a type which lacks many of the characteristics that one expects a contact-metamorphic deposit to possess. In other words, any boundary must be drawn arbitrarily; and perhaps no two people will draw it in exactly the same place. Dr. Ransome no doubt had this idea in mind in the discussion he gave to the subject in his report on the Bisbee deposits and also in his present remarks when he speaks of *typical* contact-metamorphic deposits. The question immediately arises, however, as to what is a typical contact-metamorphic deposit. Shall we regard as typical only those deposits in which the effects of igneous metamorphism have been most intense or extreme? Or shall we call typical those contact-metamorphic effects which are not so extreme and are met with more commonly? For in studying a number of districts in which contact-metamorphic deposits are involved, my associates and I have been impressed more and more with the fact that the situation in Bisbee is neither unique nor unusual.

It would appear that what is now called contact-metamorphism is simply a group of effects that stands at one end of a continuous series of effects, and that agreement must be reached as to where one shall stop in that series and say: "Contact-metamorphism ends here." Indeed, as evidence accumulates, it is somewhat doubtful in my own mind whether much is to be gained in drawing such a sharp division line. Plainly, here in Bisbee, the geologists have found no reason, in fact, have found it impossible, to draw a sharp line. It is a question in my mind whether we should not either broaden and improve our definition of contact-metamorphism in connection with ore deposits, or drop it as a significant term of practical application; for present usage is likely to lead to ambiguity and confusion and to become a cloak for loose and inaccurate thinking and expression.

THE CHAIRMAN (G. F. G. SHERMAN, Bisbee, Ariz.).—As Mr. Graton has spoken of the attitude of the companies with regard to geologists, I think we should disclaim some of the credit which he gives us. While we believe that they have been of great assistance to us, we must say that geologists did not find all the ore. We had a great deal of ore before we had geologists. A great many orebodies are found by the miners and many are stumbled on by accident. But in this complicated country, we shall always need as much assistance and information as we can get from whomever we can secure it.

Methods for Determining the Capacities of Slime-Settling Tanks*

BY H. S. COE, DENVER, COLO., AND G. H. CLEVENGER, PALO ALTO, CAL.

(Arizona Meeting, September, 1916)

ENGINEERS have long recognized the desirability of correlating the data obtained from small-scale slime-settling tests with commercial work as carried on in large tanks. This need, though most apparent in designing new installations, frequently arises also in existent plants, since a large range of experimental work can be performed without interfering with regular operation.

GENERAL SETTLING PHENOMENA

In order to develop rational methods of laboratory testing, it becomes necessary to study the general phenomena of settling. Since it was desired to give the public the benefit of these methods at as early a date as possible, together with the knowledge of the general principles of slime settling necessary for a clear understanding of the laboratory tests, we have ventured to discuss the subject in so far as brought out by the work which we now have under way. We realize that certain of our preliminary conclusions may be subject to modification, but it seems probable that, with but few exceptions, the general laws enunciated will cover the settling behavior of all pulps encountered in metallurgical plants. However, it must be recognized that there is still a vast amount of work to be done upon certain phases of the subject; therefore, at this time we will attempt only a discussion of the behavior of ore pulps during the process of settling or dewatering, under certain given conditions. The laws and principles controlling these conditions are not fully established, so that a complete discussion of them will not be attempted in this paper. In metallurgical practice, slime pulp consists of water, finely divided sand

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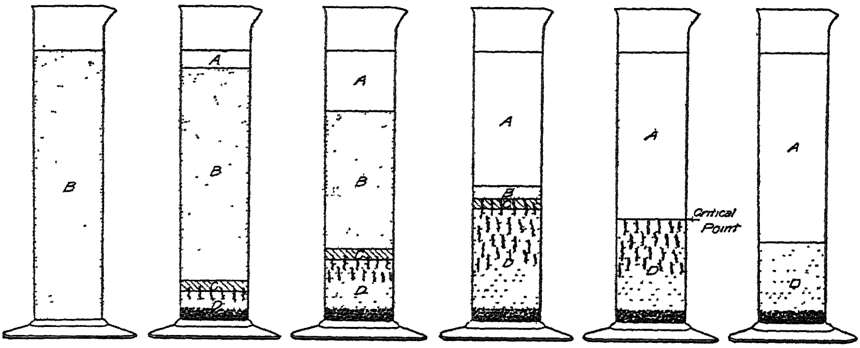
* This work was begun in the metallurgical laboratory of Stanford University and later transferred to the coöperative metallurgical laboratory of the U. S. Bureau of Mines at the Panama-Pacific Exposition. Recently Mr. Coe has continued the work on behalf of the Dorr Cyanide Machinery Co. This is contribution No. 1 from the hydrometallurgical division of the Exposition laboratory and is published with the permission of the Director of the U. S. Bureau of Mines.

or granular particles, and colloidal material. In this connection water is used as implying either ordinary water or water containing some chemical or chemicals in solution. The chemical or chemicals present which exert an influence upon the subsequent settling behavior of the pulp are known as electrolytes. The electrolyte has the property of causing the colloidal portion of the slime to form aggregates known as flocs, particles having a more or less definite size, consisting of water, colloidal material and usually fine granular material which has been entrapped. The flocs settle in the liquid medium according to certain laws.

The Four Settling Zones

If a thin pulp, of a dilution of, say, 10 to 1, is placed in a 1,000 c.c. cylinder, after thorough mixture, at least momentarily, it forms a homogeneous mass, as shown in Fig. 1(*E*). After a short time, however, it



E *F* *G* *H* *I* *J*

Fig. 1.—Experiment Showing Various Stages of Slime-Settling.

assumes a flocculent structure which, after settling a brief period of time, forms four distinct zones (*A*, *B*, *C*, and *D*), as indicated in Fig. 1 (*F*). The first particles that reach the bottom of the cylinder are the coarser granular sand which may be present in the pulp. Immediately following this and somewhat contemporaneously with the settling of the sand, the slime flocs nearest the bottom settle, filling the interstitial spaces between the sand particles, and build up, one upon another, in a zone of increasing depth. This we term zone *D*, which may be defined as that portion of the pulp wherein the flocs, considered as integral bodies, have settled to a point where they rest directly one upon another. After pulp enters zone *D*, further separation of liquid must come through liquid pressed out of the flocs and out of the interstitial spaces between the flocs. Immediately above zone *D* is a transition zone *C*. The pulp in zone *C* decreases in percentage of solids from the bottom, where the flocs enter zone *D*, to the top, where the consistency of the flocculated pulp is the

same as that of the original pulp. In speaking of flocculated pulp, it is intended to eliminate from consideration the coarser portion of the contained sand which fal's directly through the overlying zones into zone *D*. Above *C* is zone *B*, of constant consistency of flocculated pulp and of the same consistency as the flocculated pulp in the feed pulp. Zone *A*, overlying zone *B*, is clear water or solution. In the case of a very rapidly settling slime, particularly with material which has been roasted, zone *A* in the earlier stages may be turbid, due to finely divided matter remaining in suspension. Later this very fine material settles and the liquid becomes clear, although there are cases, especially when the liquid contains very little electrolyte, where it remains turbid for a long time.

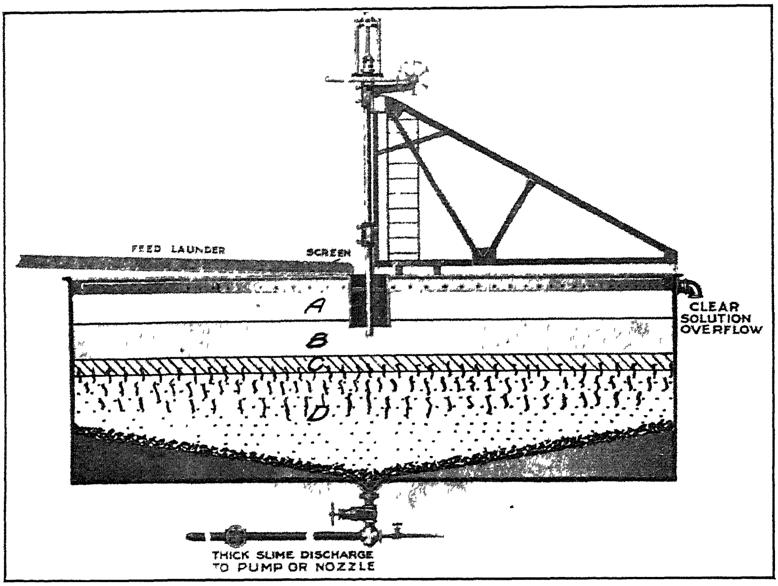


FIG. 2.—DORR THICKENER SHOWING SLIME-SETTLING ZONES.

The effect of the relations between the depths of zones *B* and *C* upon curves derived from settling tests will be discussed later.

Fig. 1 (*E*) shows a cylinder freshly filled with pulp of a consistency of about 10 parts water to 1 part ore. In this illustration zone *B* occupies the total depth. *F*, *G*, and *H* of Fig. 1 show progressive stages of settling in which zones *A* and *D* are growing deeper, zone *B* is decreasing in depth, and zone *C* remains constant—a feature of this particular pulp. Fig. 1 (*I*) shows the condition when all of the pulp has entered zone *D* and compression of the slime flocs is going on. Fig. 1 (*J*) shows the final stage of settling, beyond which the pulp will not thicken further.

With intermittent operation, any one of the stages described may represent the condition in the thickener, depending upon the length of

time that the pulp has been allowed to settle. In the operation of continuous thickeners, the feed of the thin pulp at the center of the tank, the overflow of clear liquid at the periphery of the tank, and the discharge of the thickened pulp at the bottom, are generally continuous. In a continuous thickener, the four zones previously described in discussing intermittent settling are generally present as shown in Fig. 2. At the top there is a zone of clear water, *A*. Beneath this is zone *B*, consisting of flocculated pulp of uniform consistency. Directly beneath this is a transition zone, *C*, and at the bottom a zone, *D*, of pulp which is undergoing compression.

In making tests, the settling rates of thin pulps are determined by readings taken at the juncture of zones *A* and *B*, *i.e.*, where the pulp surface joins the clear liquid.

We designate as free settling pulp all of the pulp in zones *B* and *C*, wherein the sand and flocs are falling freely through the liquid without pressing on the layers of flocs beneath, although it is evident, from the peculiarly interlocking structure of flocculated pulp, that there are points of contact between the flocs even in these zones.

Critical Settling Point Defined

We designate as the *Critical Settling Point* the top of zone *D* just as zone *C* disappears. At this point the flocs at the surface just rest upon each other, but compression has not yet commenced in the surface layer. It is therefore obvious that any elimination of liquid from zone *D* cannot be accomplished by free settling but must be effected by compression of flocs. The water liberated by compression finds its way out of zone *D* through tubes or channels which form drainage systems upward through the zone. The trunk channel for any system has its outlet at the top of zone *D*.

Since zone *B*, Fig. 1 (*E*, *F* and *G*) is made up of flocculated pulp of constant consistency,[1] the flocs in this zone will settle at a constant rate so long as the zone exists.

If zone *C* remains shallow and of constant depth, the liquid being expelled from zone *D* may be ejected through zone *C* with little admixture with this zone. This upward current of liquid which has diffused through zone *B* during the first stages of settling may also be ejected through zone *B*, when *B* becomes very shallow, and thereby considerably increase the apparent settling rate of the pulp at this stage. This does not indicate that during this period more pulp passes into zone *D*. It is merely a surface phenomenon, indicating that zone *C* is very shallow.

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[1] The constant settling rate in the upper part of the tank has been noted by Allan J. Clark in discussing the settling behavior of the slime at the Homestake. See A Note on the Settling of Slimes, *Engineering and Mining Journal*, vol. 99, No. 9, p. 412 (1915).

A curve plotted from the results of a settling test such as is illustrated in Fig. 1 will show a constant settling rate throughout the stages represented in Fig. 1 (*E*, *F* and *G*), an increased rate when the stage shown in Fig. 1 (*H*) is reached, followed by a rapid retardation in the settling rate after zone *B* disappears, until zone *C* has also disappeared. Following this is a very slow rate of settling which gradually becomes slower as the water is compressed from zone *D*. The final stage of settling is reached when no further liquid is expelled from zone *D* by compression. The later stages of settling are shown in Fig. 1 (*I* and *J*).

Slimes Classified as to Settling Rate

The type of settling showing a constant rate down to near the critical point, regardless of whether acceleration is manifested or not, we term

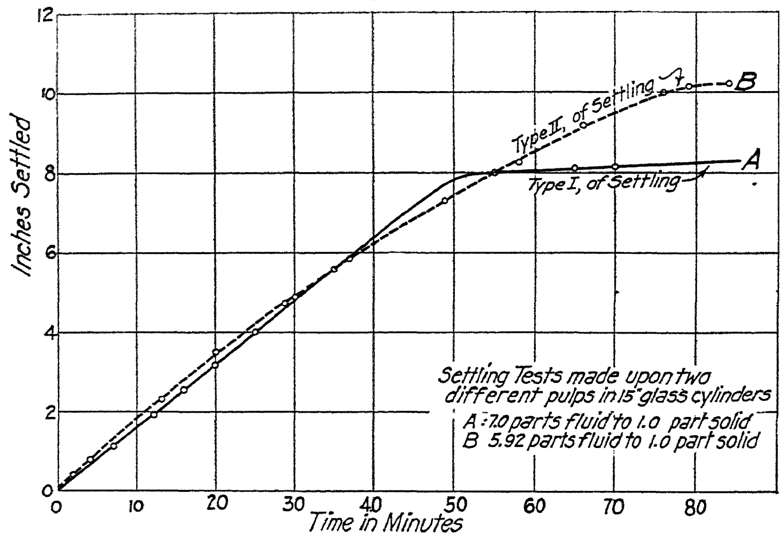


FIG. 3.—CURVES SHOWING SETTLING RATES OF TWO TYPES OF SLIME.

Type I. This is illustrated in Fig. 3 by curve *A*. With certain pulps a constant rate of settling does not occur, since zone *C* rapidly reaches the surface. The surface of such a pulp contained in a cylinder, therefore, settles at a continually decreasing rate down to the critical point. We term this type of settling Type II. It is illustrated by curve *B*, Fig. 3.

Type I of Settling

A pulp containing very little sand was mixed with water containing 40 points of lime (100 points being equivalent to a saturated solution of lime in water, *i.e.*, 0.13 per cent.) to give a pulp of a consistency of 5.7

parts of liquid to 1 part of ore. This pulp settled at a constant rate to near the critical point. Tests were also made to ascertain the settling rates of pulps of 4.71 and 4.00 parts of liquid to 1 part of ore, respectively.

	Consistency of Pulp Parts Water to 1 Part Ore	Rate of Settling Feet per Hour
I.	5.7	0.6
II.	4.71	0.464
III.	4.00	0.414

Conceive a column of pulp consisting of three distinct layers: I (upper), II (middle), and III (lower), having consistencies of 5.7, 4.7 and 4.00 parts fluid to 1 part solids, respectively, layer I feeding into layer II, and layer II feeding into layer III. Let it be assumed that the liquid contained in the flocs and being carried down with them is 3 parts to 1 part of solids. Since the flocs in the three layers are passing downward at known rates, we may compute the weight of solids being fed from a given area of each layer per unit of time, *i.e.*, from layer I to layer II, from layer II to layer III, and leaving layer III, as follows: The pulp in layer I consists of 5.7 parts fluid to 1 part of solids, making a total of 6.7 parts. Of this, 3 parts fluid and 1 part solids are passing downward while 2.7 parts of fluid remain behind. Therefore, for each 2.7 parts of fluid which remain behind, 1 part of solids passes downward and out of layer I. Since the pulp in layer I settled at the rate of 0.6 ft. per hour, the water remaining behind in an area of 1 sq. ft., in 1 hr. would be 0.6 cu. ft., equivalent to 37.41 lb. (water weighs 62.35 lb. per cubic foot) and the weight of solids passing out of layer I per hour would be 13.8 lb. per square foot. In a similar manner it is calculated that the solids passing out of layer II are 16.9 lb. per square foot per hour and the solids passing out of layer III are 25.8 lb. per square foot per hour. The following table shows the weight of solids, in pounds, which may be fed from each layer, assuming the quantity of fluid carried downward in the flocs to be as follows: 1 part fluid to 1 part solids; 2 parts fluid to 1 part solids; 2½ parts fluid to 1 part solids: and 3 parts fluid to 1 part solids.

Layer	1 to 1 Pounds	2 to 1 Pounds	2½ to 1 Pounds	3 to 1 Pounds
I.	7.95	10.1	11.6	13.8
II.	7.78	10.6	13.1	16.9
III.	8.60	12.9	17.2	25.8

It will be noted that if the flocs are assumed to contain between 2 to 1 and 3 to 1 parts of liquid, the weight of solids passing out of layers II and III will be greater than the weight passing into each from the overlying layer. Therefore, layers II and III must decrease in depth, and finally become infinitely shallow. The rate of settling will therefore

remain constant in a settling test on the pulp of the consistency of 5.7 parts liquid to 1 part solids, until the pulp becomes thicker than that in layer III. If the flocs contained only 1 part liquid to 1 part solids, layer II would increase in depth and in a continuous settling test the rate of settling would decrease until the rate of 0.464 ft. per hour was reached and thereafter would remain constant until all of the pulp became thicker than that in layer III. Since investigation shows that the settling rate remains constant almost to the critical point, the proportion of liquid in the flocs must be greater than 1 to 1.

Type II of Settling

Fig. 3 (*B*) is the curve plotted from the results of a continuous settling test made upon a pulp showing Type II[2] settling. This pulp showed a very fine flocculated structure. The consistency was 5.92 parts fluid to 1 part solids. It was placed in a 1,000 c.c. cylinder and allowed to settle without interruption to the critical point. Readings were taken at intervals and the settling rates in feet per hour were computed.

After permitting the sand to settle out of a sample of this pulp, it was found that the free-settling sand represented 46 per cent. of the total solids. Settling rates were determined on pulps of varying consistencies. The consistencies are estimated as the ratio of fluid to total solids (1) and as the ratio of fluid to flocculated solids (2). The consistencies and settling rates are given below:

Layer	Ratio of Fluid to Total Solids (1)	Ratio of Fluid to Flocculated Solids (2)	Rate of Settling in Feet per Hour
I.	5.92–1	10.95–1	1.03
II.	5.17–1	9.57–1	0 89
III.	4.42–1	8.20–1	0.69
IV.	3.67–1	6.79–1	0 52
V.	3.17–1	5.87–1	0.417
VI.	2.92–1	5.40–1	0 372
VII.	2.42–1	4.48–1	0 297
VIII.	2 00–1	3.70–1	0 248

Using the same line of reasoning that was followed in estimating the settling capacities through the various layers in discussing Type I of settling and estimating the amount of fluid contained in the flocs at 2 parts fluid to 1 part flocculated slime and also at 3 parts fluid to 1 part flocculated slime, the following capacities are obtained in pounds per hour of flocculated slime which may pass out of each respective layer:

[2] The same ore may yield both Type I and Type II of settling under varying conditions.

Layer	Flocs Containing 2 to 1 Pounds	Flocs Containing 3 to 1 Pounds
I.	7.16	8.06
II.	7.32	8.43
III.	6.93	8.26
IV.	6.75	8.55
V.	6.70	9.00
VI.	6 81	9.63
VII.	7.50	12.58
VIII.	9.12	22.40

If we assume that the flocs are made up of 3 parts fluid to 1 part flocculated solids, the estimated weight of solids passing out of each layer appears to be greater than the amount fed into it from above. In this event the lower layers could not exist and the settling would, therefore, follow Type I. If we assume flocs made up of 2 parts fluid to 1 part solids, the estimated weight of solids passing out of each layer (eliminating the settling rate in II, which was probably erroneous) into the layer below becomes less than the amount fed into it down to layer VI, after which the quantity of solids passing out of each layer is greater than the weight fed into it from above. In this case layers V, IV, III, etc., would increase in depth as layers I, II, etc., disappear, and the settling rate indicated by a continuous settling test made in a cylinder, filled with pulp similar to that in layer I, will decrease continuously until the pulp at the surface is of a consistency between that of the pulp in layers V and VI. Layers VI to VIII will be infinitely thin, since it is possible for more pulp to pass out from each layer than the amount which enters it and, therefore, the settling rate of pulps of the consistencies contained in these layers will only appear in a period of rapid transition from the settling rate in layer V to the settling rate in some layer below VIII. It has been calculated, from the data from which curve *B* (Fig. 3) was plotted, that the rate of settling decreased in 66 min. from 1.03 ft. per hour, as in layer I to 0.55 ft. per hour, and in the next 18 min. dropped to 0.08 ft. per hour. From the above it should be evident that the weight of solids which can pass from each layer is dependent upon the percentage of sand present (which passes downward carrying no liquid), upon the ratio of fluid to solids in the flocs and upon the settling rate of the pulp in the layer considered. The capacity of a layer of any consistency of flocculated pulp to discharge its flocculated solids may be calculated by the following formula:

$$c = \frac{62.35r}{f - d}$$

c = capacity in pounds of flocculated solids per square foot per hour.
r = settling rate in feet per hour in the layer.

f = ratio of fluid to flocculated solids in the layer.
d = ratio of fluid to flocculated solids in the flocs.
62.35 = weight of 1 cu. ft. of water.

In the illustration of Type II of settling, pulp of a consistency of 3.17 parts fluid to 1 part solids or 5.87 parts fluid to 1 part flocculated solids settled at a rate of 0.417 ft. per hour. Assuming that the flocs contain 2 parts fluid to 1 part solids, the above formula gives $\frac{62.35 \times 0.417}{5.87 - 2.0} = 6.7$ lb. per square foot per hour of flocculated solids and 13.4 lb. of fluid passing downward out of layer V, but since the flocculated solids represent only 54 per cent. of the total solids, the total weight of solids passing downward out of layer would be 12.4 lb. per square foot per hour. The proportion of fluid carried down in the flocs to total solids would be 13.4 : 12.4 :: 1.08 : 1. Given the ratio of fluid to total solids in a pulp with a known settling rate and the ratio of fluid to total solids to be discharged, it is not necessary to know the percentage of fluid in the flocs in order to determine the maximum capacity of such a layer of pulp to discharge pulp of the consistency required. Thus in the above case the ratio of fluid to total solids equals 3.17 to 1, and the ratio of fluid to solids in the discharge required is 1.08 to 1. Applying the formula, we find $c = \frac{62.35 \times 0.417}{3.17 - 1.08} = 12.4$ lb., which is the same as that given above. The formula may, therefore, be written $C = \frac{62.35R}{F - D}$, where

F = ratio of fluid to solids in the pulp tested.
R = rate of settling in feet per hour of a free-settling pulp of consistency F.
D = ratio of fluid to solids in the discharge required.
C = capacity in pounds of solids per square foot per hour, which may be discharged with a consistency of D from a layer of pulp of a consistency of F settling at a rate R.

Application of the Formula in Determining the Capacity of Continuous Thickeners

Since in any vessel used for settling there are layers of pulp of every consistency between that of the feed and that of the discharge, although some of the layers may be infinitely shallow, the maximum capacity in pounds of solids per square foot per hour is expressed by the formula $C = \frac{62.35R}{F - D}$, and the maximum capacity possible will be the smallest value of C obtained by applying this formula to a series of tests made upon pulp ranging in consistency from that of the feed to that of the

thickest free-settling pulp, and taking for D the ratio of fluid to solids desired in the discharge.

Thickening in Zone D and Consistency of Discharge Possible

We have established through numerous experiments the fact that after pulp reaches the consistency where the flocs touch each other, further elimination of water becomes approximately a function of time, in so far as tests for metallurgical practice are concerned. It seems probable that the relationship of pulp consistency to time of thickening is an indirect one, depending upon the effect of compression caused by the depth of the pulp being counteracted by the resistance in the pulp to the escaping water, and to the admixture of the pulp in the upper portion of the thickening layer with the ascending water from the lower layers. A large number of comparative tests have been made in vessels of from 1 to 10 ft. in depth. If portions of pulp be placed in vessels of various depths, it will be found that the critical point will be reached earliest by the surface of the pulp in the most shallow vessel, since the pulp does not have so far to settle to reach the zone of compression (D), but the settling rates in layers of like consistency are the same for vessels of all depths. In the deep vessels the critical point will be proportionately lower than in the shallow vessels. This may be explained by the fact that the average time of compression in the thickening zone before the critical point is reached is greatest for the pulp in the deep vessel. If the fluid be expelled as a function of time after the flocs enter the zone of compression, it is but natural that this should be the case.

In two vessels of unequal depth, filled with the same kind of pulp and started to settle at the same time, the total pulp in the more shallow vessel will begin to compress more quickly and will thereafter for many hours remain thicker than the pulp in the deeper vessel, even though the critical points occur not more than an hour apart. This indicates that thickening in the compression zone is a function of time. In one or two cases it was noted that after a time pulp in the deeper vessel became the thickest, but, upon decanting the fluid from the pulp contained in the shallow vessel and stirring the remaining pulp, settling was resumed until the pulp was as thick or thicker than that in the deeper vessel. We assume that this difference was due to the fact that most of the sand in the shallow vessel reached the bottom, while in the deeper vessel some of the sand lodged in the lower part of the compression zone and assisted compression.

Another series of tests was made by placing moderately thick pulp in cylinders of various depths. It was found that after allowing time for the structure to develop and channels to form in the deep cylinder, the total settling at various periods of time was approximately propor-

tional to the depth of the cylinder. There is, however, a limit to this, for it appears that as the settling by compression in a deep vessel reaches the slowest settling rate for free-settling pulp, a zone of semi-thickened pulp will be formed and maintained until the compression rate for the total column slows down to a lower rate than the lowest free-settling rate of the pulp. In this series of tests the greatest consistency for various periods of time is almost invariably reached in the most shallow cylinders, but the difference is slight and to a great extent due to the resistance of the pulp to liquid rising from the bottom of the cylinder. In practice in continuous thickening, such a zone of semi-thickened pulp never reaches the bottom of the tank, since large amounts of fluid do not have to be liberated from the pulp at the bottom. This resistance is therefore of less consequence in practical operation.

The diameter of the cylinder or tank used for making the test seems to have little or no influence upon settling. Tests made upon various pulp samples in cylinders varying in diameter from ½ in. to 2½ in. show very slight differences in the settling rates and in the final consistency of the settled pulp. The ½-in. tube generally gave a more rapid rate of settling down to the critical point, probably due to convection currents. Beyond this point, the pulp in the tube of small diameter settles more slowly. The following experiment will illustrate this point: A 4-ft. cylinder, 2½ in. in diameter, and a 4-ft. cylinder, ½ in. in diameter, were filled with Liberty Bell pulp, 3.26 parts water to 1 part dry slime. The water contained 4 points of lime. The consistency of the settled pulp, after settling 48 hr. in the 2½-in. cylinder was 1.8 parts water to 1 part of dry slime, while in the ½-in. cylinder it was 1.825 parts water to 1 part dry slime. It was found that the difference was greater when the water contained a greater proportion of electrolyte (in this case, lime), for the reason that as the proportion of electrolyte is increased, the slime flocs become larger and agglomerate in such a manner as to offer more resistance to settling in the small cylinder. With a large proportion of lime, the pulp in the small cylinder settled in divisions, due to the arching effect of the large slime flocs, there being alternate layers of clear liquid and of thick pulp. It was noted that ultimately the sum of the zones of clear liquid equaled the total depth of clear liquid formed in the larger cylinder. All of our tests have indicated that if the diameter of the cylinder is 2 in. or over, little or no influence is exerted upon the rate of settling.

Table I gives the record of some of the tests made to establish the fact that ultimate consistency of the pulp is a function of the time during which the pulp remains in the thickening zone. The names of the mines from which these pulps came are not given, as permission to publish them was not obtained. Example *G* in Table I was made on a pulp consisting of 6 parts solution of zinc sulphate containing hydrogen sul-

phide to 1 part precipitated barium sulphate and zinc sulphide. Despite the low percentage of solids, the pulp is very viscous and is not free settling. Air bubbles remained entrapped for 36 hr. when the pulp was allowed to settle in a cylinder 16 in. in depth. In a cylinder 56 in. in depth the bubbles were ejected and the settling rate became 50 per cent. greater than the ratio of the depth to that of the lower cylinder would indicate. It would seem that in this type of pulp the law of compression being a function of time would not hold. Apparently the reason is that the highly viscous homogeneous structure resists the formation of tubes and channels in zone *D*, except under higher pressure through a deeper column of pulp. It is possible that with a large proportion of electrolyte, for instance, lime, certain ore pulps also may behave in this manner. When such cases arise, recourse must be had to the test described later in the paper. Such a condition is generally indicated by the absence of the ordinary flocculated structure and the fact that active channeling does not take place in zone *D*.

Table I.—*Comparative Settling after Equal Intervals of Time in Cylinders of Various Heights*

Example A

Type of Pulp Siliceous; 60 Per Cent. through a 200-mesh Screen

Time of Settling, Hours	Height of Cyl. 91.25 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 44.75 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 14.3 In Settling in Inches	Consistency of Pulp, Fluid to Solid
0	0 00	2.95–1	0.00	2.95–1	0 00	2.95–1
1	4.25	....	3.92	....	3 60	
2	9.00	....	9.22*	....	4 76	
4	19.20*	....	14.70	....	5.90*	1 58–1
19	37.90	1.56–1	19.50	1.50–1	7.80	1.14–1
24	39.90	1.50–1	20.40	1.43–1	....	
28	41.80	1.42–1	20.40†	1.43–1	....	
44	43.75	1.38–1	22 60	1.27–1	7.80	1.14–1
	* Tubes to the surface.		* Tubes to the surface. † Drew off water and stirred.		* Drew off water and stirred.	

Example B

Time of Settling, Hours	Height of Cyl. 29.75 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 3 In. Settling in Inches	Consistency of Pulp, Fluid to Solid
0	0.00	3.00–1	0.00	3.00–1
16	....	....	1.56	1.18–1
24	17.37	1.05–1	1.78	1.00–1
32	18.50	0.90–1	1.83	0.95–1

TABLE I.—*Comparative Settling after Equal Intervals of Time in Cylinders of Various Heights*—(*Continued*)

Example C

Siliceous Ore

Time of Settling, Hours	Height of Cyl. 123 In. Settling in. Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 14.3 In. Settling in Inches	Consistency of Pulp, Fluid to Solid
0	0	4.73–1	0.0	4.73–1
18	113	1 05–1	13.15	1.08–1
	....	....	Removed H_2O and stirred.	
36	....	....	13.37	0.90–1

Example D

Very Colloidal Siliceous Pulp

Time of Settling, Hours	Height of Cyl 92 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 41.5 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 14.1 In. Settling in Inches	Consistency of Pulp, Fluid to Solid
0	0.0	2.13–1	0.0	2.13 –1	0.00	2.13–1
17	30.3	1.30–1	14.8	1.235–1	5.57	1.13–1
20	31.0	1.28–1	15.1	1.219–1	5.72	1.11–1
64	36.3	1.13–1	18.4	1.020–1	6.64	0.95–1

Example E

Liberty Bell Pulp Oxidized with a High Percentage of Clay; CaO Traces

Time of Settling, Hours	Height of Cyl 114 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl 45 In. Settling in Inches	Consistency of of Pulp, Fluid to Solid	Height of Cyl. 11 In. Settling in Inches	Consistency of Pulp, Fluid to Solid
0	....	3.26–1	....	3.26–1	....	3.26–1
1	1.5	....	1.5	....	1.56	2.75–1
5	9.4	2.95–1	9.75	2.47–1	3.92	1.97–1
23	45.5	1.81–1	19.5	1.69–1	5.25	1.53–1
29	51.5	1.62–1	21.0	1.56–1	5.39	1.49–1

Example F

Enterprise Ore—Siliceous

Time of Settling, Hours	Height of Cyl. 60 In. Settling in Inches	Consistency of Pulp, Fluid to Solid	Height of Cyl. 10 In. Settling in Inches	Consistency of Pulp, Fluid to Solid
0.00	0.00	7.00–1	0.00	7.00–1
0.50	4.63	....	4.50	3.68–1
3.00	24.43	....	6.18	2.43–1
4.75	35.78	2.61–1	6.50	2.20–1
18.00	41.27	1.93–1	7.10	1.76–1

TABLE I.—*Comparative Settling after Equal Intervals of Time in Cylinders of Various Heights—(Continued)*

Example G

A Chemical Pulp, Showing an Apparent Exception

Time of Settling, Hours	Height of Cyl 56 In. Settling in Inches	Height of Cyl. 15.75 In. Settling in Inches	Consistency of Pulp, Fluid to Solid
0	0.0	0.0	6–1
7	5.9	1.2	
24	11.3	2.5	

SUMMARY OF SLIME-SETTLING RESULTS

The results of our investigation of slime settling may be summarized as follows:

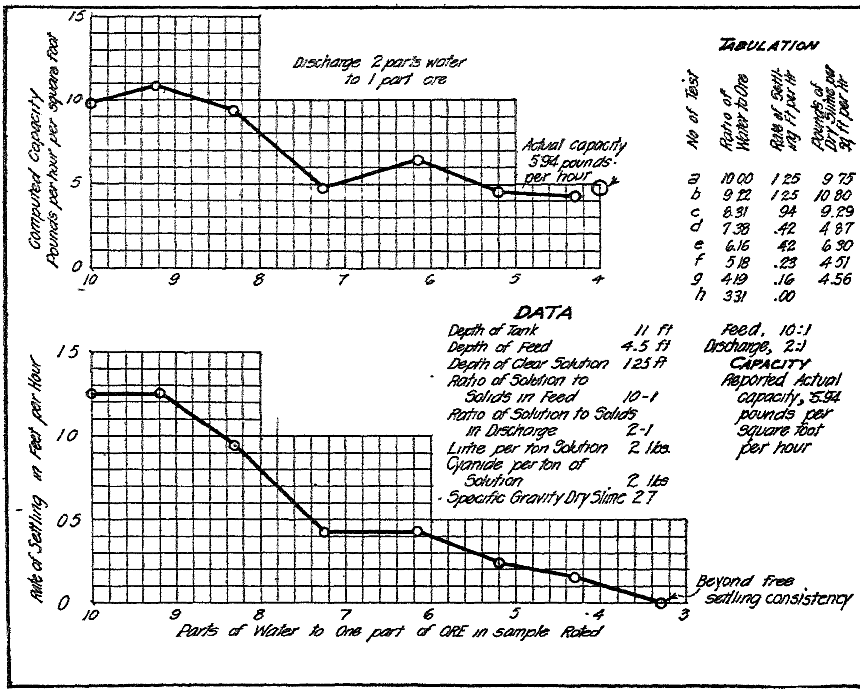


FIG. 4.—SLIME-SETTLING DATA, LIBERTY BELL MILL, TELLURIDE, COL.

1. In thickening pulps which are to be discharged at a consistency such that the discharge is still in the form of free-settling pulp, the depth of the tank used is of no consequence, except in so far as it permits a depth of feed sufficient to avoid surface agitation and allows ample depth of clear liquid to care for fluctuations of the feed and changes in

the character of the pulp in the case of continuous thickeners, and sufficient depth to give ample capacity to avoid the necessity of frequent charging and discharging in the case of intermittent thickeners.

2. When thickening pulps to a consistency where it is necessary to expel fluid by compression, sufficient capacity must be given the tank so that the pulp will be retained the necessary period of time to thicken it to the required density and also to allow sufficient storage to compensate for fluctuations in the feed and discharge. This capacity may be obtained by varying either the depth or diameter to give the required cubical content.

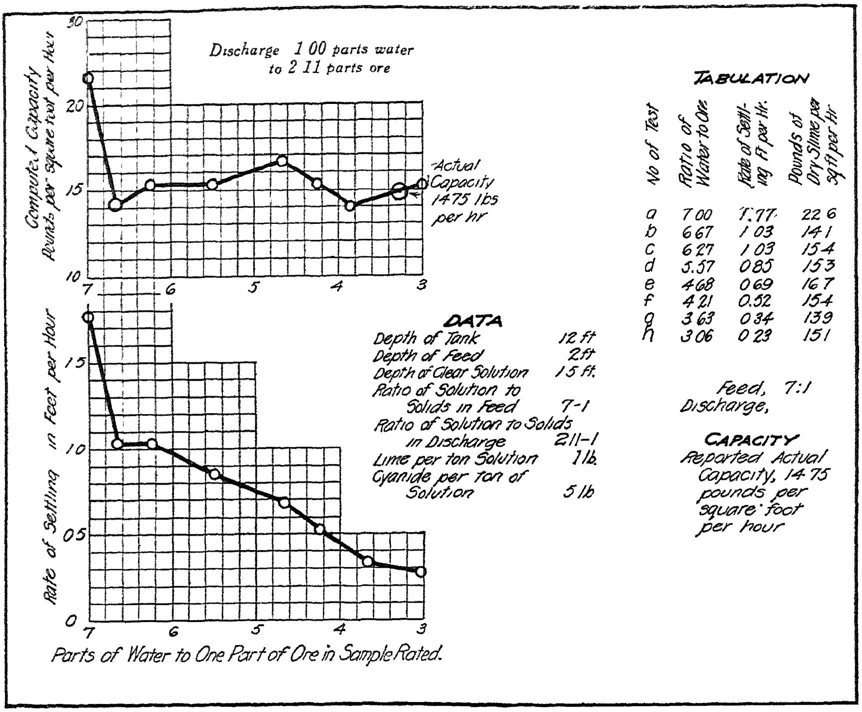


Fig. 5.—Slime-Settling Data, Belmont Mill, Tonopah, Nev.

3. The consistency of discharge possible may be closely determined by allowing a cylinder of thick but free-settling pulp to settle, taking readings at intervals of a few hours up to the point where settling practically ceases.

4. The required area may be computed by applying the formula $A = \dfrac{2,000}{24\left(\dfrac{62.35R}{F-D}\right)}$. A is the area in square feet required to thicken 1 ton of 2,000 lb. of solids to a consistency in the discharge of D (parts fluid

to 1 part solids by weight), per day of 24 hr. A series of settling rates is taken on pulps ranging in consistencies from that of the proposed feed to the thickest free-settling pulp. D is taken as the ratio of fluid to solids in the thickest pulp which can be economically obtained. The greatest value for A indicated by the tests is taken as the required area. Under ordinary circumstances a factor of safety should be allowed over the calculated area to take care of changes in the character of the pulp and variations in temperature. It will be noted that this is the same formula previously

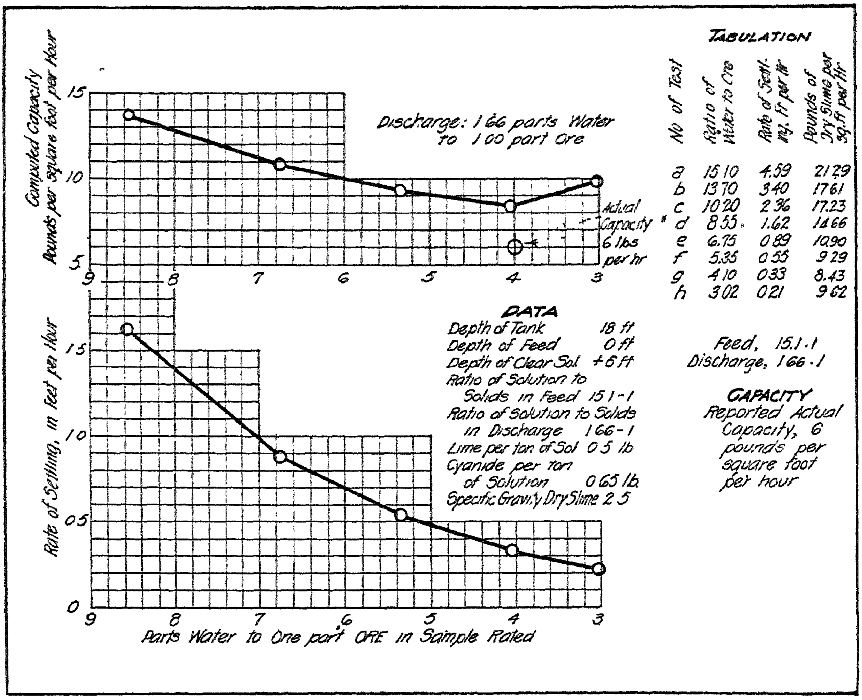


FIG. 6.—SLIME-SETTLING DATA, PORTLAND MILL, VICTOR, COL.

given, modified to give area required instead of capacity per square foot per hour.

5. The required depth of the thickener may be ascertained by computing the capacity of the thickening zone to contain a supply of solids equal to the total capacity of the tank for the number of hours required to thicken the pulp to the density required in the discharge, and to this depth adding an allowance for the lost space due to the pitch of the drag in the thickener; also from 1½ to 2½ ft. for depth of feed and a further allowance for storage capacity when the discharge may be closed. The following is an illustration of the method used in computing the area and depth required in a thickener to handle 100 tons of pulp per day, thicken-

ing from 6 parts fluid to 1 part solids down to 1.12 parts fluid to 1 part solids.

Test	Parts Fluid to 1 Part Solids	Rate of Settling in Feet per Hour	Area Required per Ton of Solids per 24 Hours
1	6.00	2.180	3.00
2	4.94	1.190	4.29
3	4.00	0.893	4.31
4	3.51	0.758	4.22
5	3.00	0.600	4.05

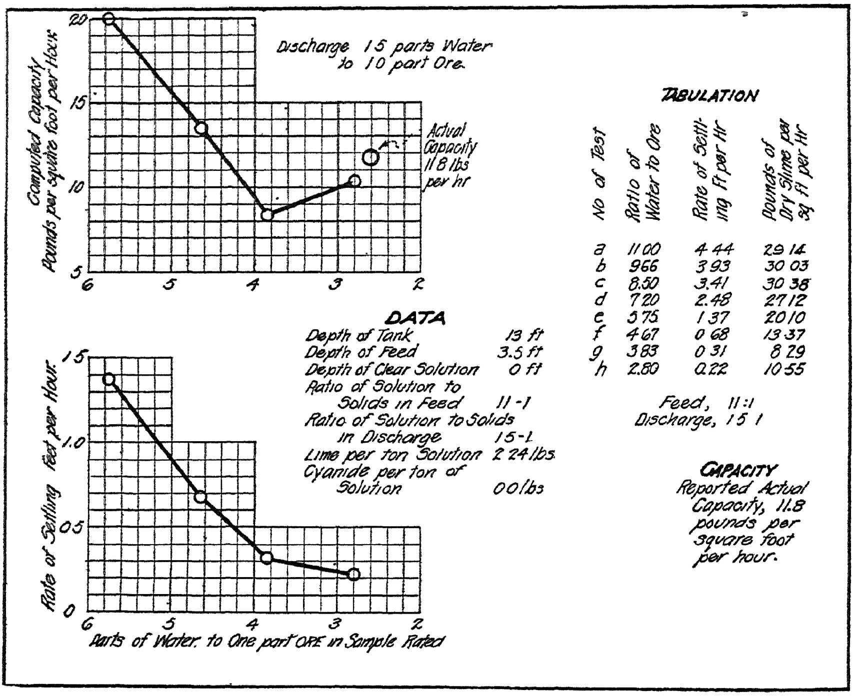


FIG. 7.—SLIME-SETTLING DATA, NIPISSING LOW-GRADE MILL, COBALT, ONT.

The formula $A = \dfrac{2{,}000}{24\left(\dfrac{62.35R}{F-D}\right)}$ was applied to ascertain the above areas. Taking for instance the third test, $R = 0.893$, $F = 4$, $D = 1.12$, then $A = \dfrac{2{,}000}{\dfrac{24 \times 62.35 \times 0.893}{2.88}} = 4.31$, the number of square feet required per ton per day. For handling 100 tons per day, 431 sq. ft. or the area of a tank 23.4 ft. in diameter would be required. In the above example the third test is taken for the required area as being the largest area indi-

cated by any of the tests, while the areas indicated by the fourth and fifth tests show that the areas are diminishing for the thicker pulps, therefore the turning point has been passed.

To Ascertain the Necessary Depth of Tank

Assuming that the following results are obtained from settling tests made in a cylinder 12 in. deep upon 3 to 1 pulp, the computations are made as follows:

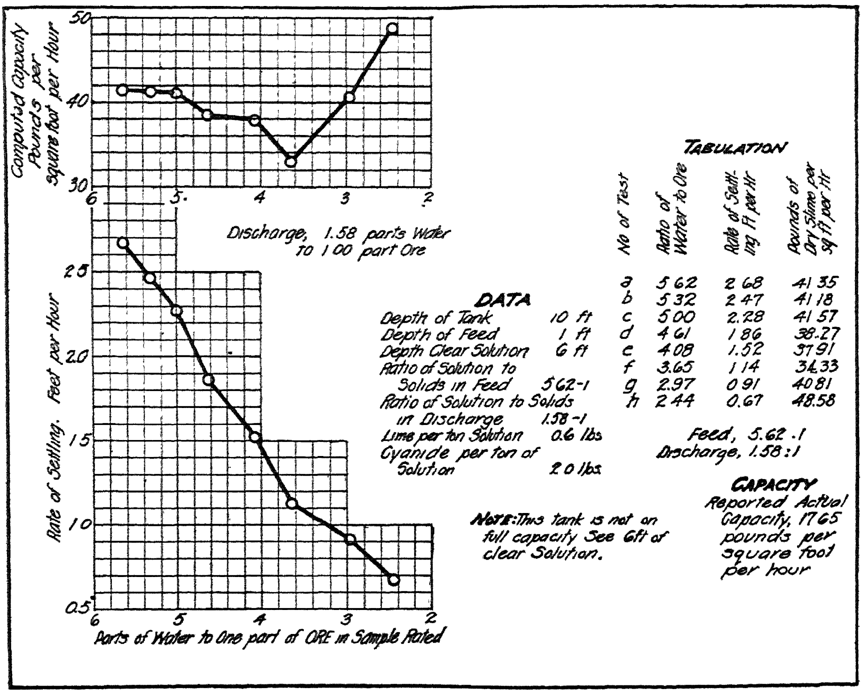


FIG. 8.—SLIME-SETTLING DATA, PRESIDIO MILL, SHAFTER, TEX.

Time of Thickening Hours	Consistency Fluid to Solids
2	1.70 :1
4	1.59 :1
9	1.35 :1
14	1.20 :1
19	1.12 :1

Since an area of 4.31 sq. ft. is required per ton of solids per 24 hr., the total solids per square foot retained in the thickening zone must be $\frac{19 \times 2{,}000}{24 \times 4.31} = 367$ lb., or 19 hours' supply per square foot. There is required a 5-hr. supply, of each of the pulp consistencies 1.16 to 1; 1.275

to 1; 1.47 to 1, and a 4-hr. supply of a 1.7 to 1 pulp. The solids per cubic foot in the above pulps are 43.2 lb., 37.6 lb., 33.7 lb. and 30 lb., respectively. The depth of each class of pulp would therefore be 2.23 ft., 2.57 ft., 2.87 ft., and 2.58 ft., or a total depth of 10.25 ft. To this depth must be added a foot for the loss due to the pitch of the drags in the thickener and 1.5 ft. for the depth of the feed, since the feed is thick and the volume will be proportionately low. The total calculated depth of the tank required would be 12.75 ft. If proper allowance were made for storage capacity, the tank might be inconveniently deep. In this case it would be better practice to make the tank 12 ft. deep and make

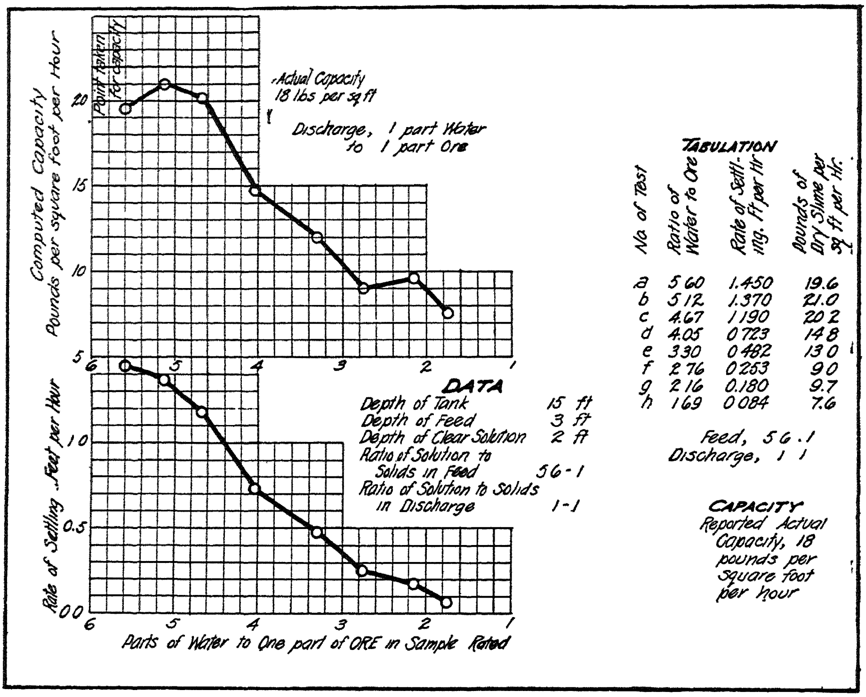


FIG. 9.—SLIME-SETTLING DATA, HOLLINGER MILL, TIMMINS, ONT.

various allowances for additional capacity by increasing the diameter of the tank to give a 30 per cent. increase in area. Ordinarily in settling ore pulps composed of fine granular material with a considerable proportion of colloidal material which may vary in character, it is unwise to provide less than 6 sq. ft. of settling area per ton of solids to be settled daily. Frequently much larger areas are required.

If, in the tests previously cited, the fourth and fifth settling rates had decreased so as to cause the indicated required areas to increase gradually, the solution of the problem would be less simple, for it is necessary to be certain that the settling rate for the thickest free-settling

pulp has been determined and also that the pulp being tested is free settling. Ordinarily the latter point is indicated by the evenly flocculated appearance of the pulp surface, without channels of fluid coming through, and by the uniform texture of the pulp as seen in a glass cylinder. The settling rate, after flocculation is complete, should be constant or gradually diminishing to the critical point. In certain thick, free-settling pulps containing a large proportion of electrolyte, the whole mass agglomerates upon shaking, and in a shallow cylinder the time required to form the natural structure will be so great that the pulp will have passed the critical point before a reliable settling rate has been observed.

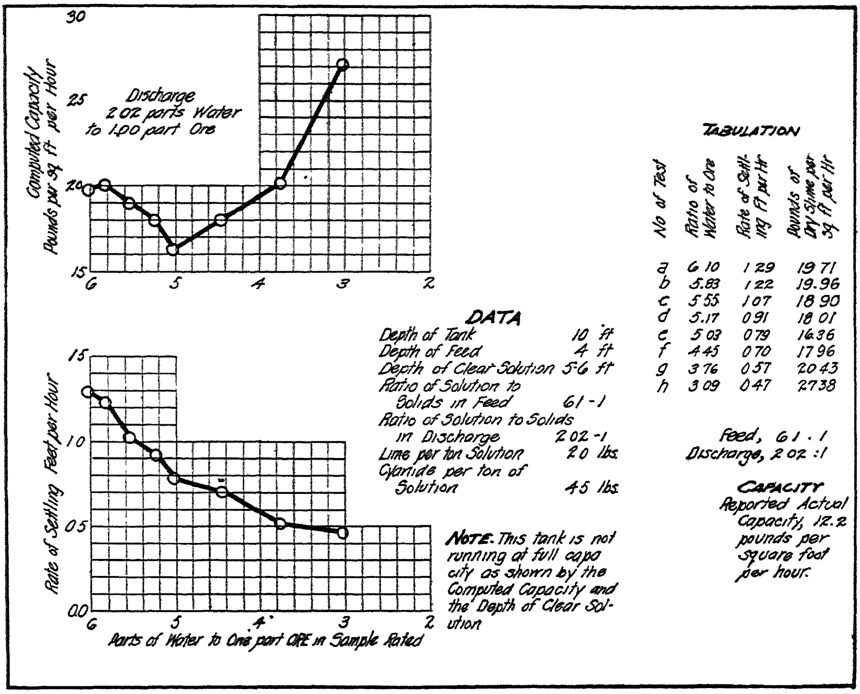


FIG. 10.—SLIME-SETTLING DATA, WEST END CONSOLIDATED MILL, TONOPAH, NEV.

In such cases the series of tests should be repeated using less electrolyte, the object being to ascertain the consistency at which the indicated area establishes itself as a constant, or decreases. Should it not be feasible to reduce the proportion of electrolyte, cylinders 3 or 4 ft. in height may be employed in repeating the tests, in order that sufficient time may be given for the flocculated structure and even texture to become apparent. If, after applying these tests, it is desired to verify the results still further, it becomes necessary to make a test representing actual continuous feed and discharge conditions, as hereafter described in this paper.

In making settling tests the sample should be carefully taken in order that it may be truly representative of the pulp to be settled. The ratio of fluid to solids may be determined by drying a sample from each test. The pulp is placed in a 1,000 c.c. cylinder and well shaken; it is then allowed to settle. Readings should be taken at intervals of from 1 to 3 min. until the pulp has reached a uniform or decreasing settling rate, the maximum rate being taken as the one sought. The readings are most conveniently taken in cubic centimeters and the settling rate in feet per hour may be computed from these by establishing the value

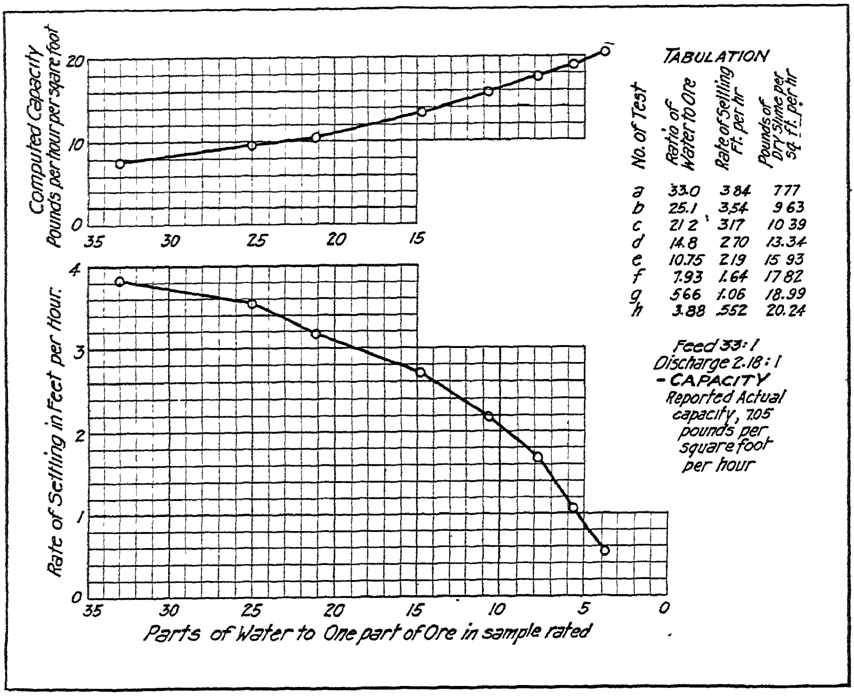


FIG. 11.—SLIME-SETTLING DATA, HOMESTAKE MILL, LEAD, S. D.

in decimal parts of a foot of each cubic centimeter. This is done by measuring, in feet, the length of the 1,000 c.c. graduation on the cylinder and dividing by 1,000.

METHOD OF MAKING SETTLING TESTS

The method of making the settling tests hereafter quoted was as follows: In each case the pulp was permitted to settle ⅛ in. before the readings were taken.

(a) 1,000 c.c. feed pulp settled for a 2-min. interval.

(b) After completing test (a) and thoroughly mixing by shaking,

25 c.c. of pulp was removed from the cylinder and replaced by 25 c.c. of discharge pulp. After thoroughly mixing, allowed to settle for an interval of 2 min.

(*c*) 30 c.c. of pulp removed from (*b*) and replaced by 30 c.c. of discharge pulp, settling interval 2 min.

(*d*) 45 c.c. of pulp removed from (*c*) and replaced by 45 c.c. of discharge pulp, settling interval 3 min.

(*e*) 75 c.c. of pulp removed from (*d*) and replaced by 75 c.c. of discharge pulp, settling interval 3 min.

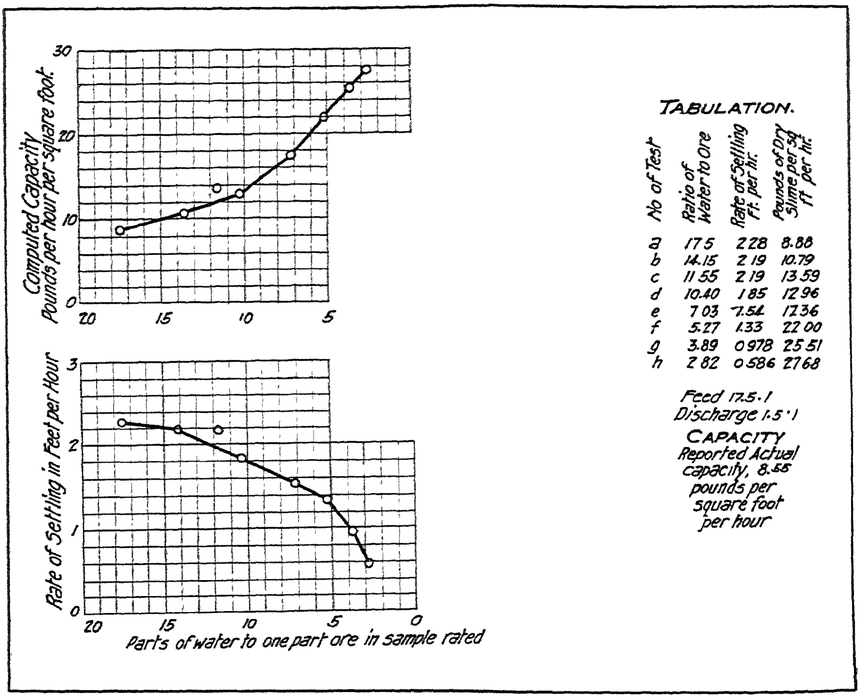


FIG. 12.—SLIME-SETTLING DATA, HOMESTAKE MILL, LEAD, S. D.

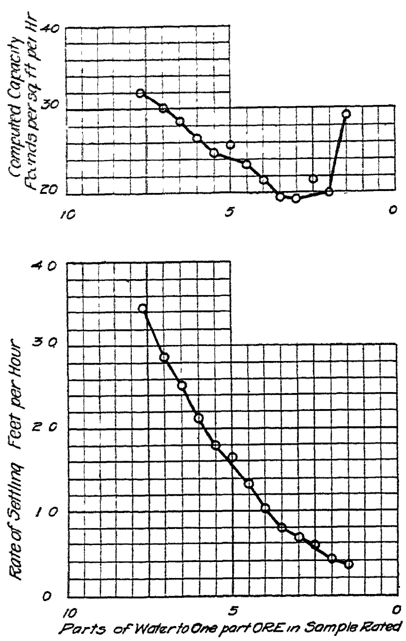
(*f*) 100 c.c. of pulp removed from (*e*) and replaced by 100 c.c. of discharge pulp, settling interval 4 min.

(*g*) 160 c.c. of pulp removed from (*f*) and replaced by 160 c.c. of discharge pulp, settling interval 4 min.

(*h*) 260 c.c. of pulp removed from (*g*) and replaced by 260 c.c. of discharge pulp, settling interval 6 min.

Throughout a series of tests the precaution must be taken of thoroughly mixing the contents of the cylinder both before removing pulp and before starting the test on the pulp of altered consistency.

The curves in Fig. 4, showing the settling behavior of Liberty Bell pulp, illustrate the method of plotting used in presenting the results of the tests which have been taken as a basis of comparison with the performance of commercial thickeners. In the lower curve the settling rate in feet per hour is taken as the ordinate, while the ratio of water to dry slime is taken as the abscissa. Each point shown on the curve represents an individual test made upon slime of the dilution noted. The tests appear upon the curve from left to right in the order performed, which places the highest dilution at the origin.



TABULATION

No of Test	Ratio of Water to Ore	Rate of Settling Ft per Hr	Pounds of Dry Slime per sq ft per Hr
a	7 7	3 44	32 0
b	7 0	2 89	30 0
c	6.5	2.52	28 6
d	6.0	2.11	26 3
e	5 5	1 80	24 9
f	5 0	1 66	25 9
g	4 5	1.31	23 3
h	4 0	1.04	21 6
i	3 5	785	19 6
j	3 0	.62	19 3
k	2 5	525	21 8
l	2.0	.320	20 0
m	1.6	280	29 1

Feed, 7.7:1
Discharge, 1:1

CAPACITY
Computed capacity 19 3#
Reported capacity on long test on 6' experimental thickener 19#
DIFFERENCE = 1 57%

FIG. 13.—SLIME-SETTLING DATA, GOLDEN CYCLE MILL, COLORADO SPRINGS, COL.

In the upper curve, the computed capacity in pounds per hour of dry slime per square foot of settling area is taken as the ordinate, while the dilution is taken as the abscissa, as in the lower curve.

The following table gives the actual capacities, together with the computed capacities, of thickeners used at the various mills enumerated, as shown in curves, Figs. 4 to 13, inclusive. In each case the tests were made at the mill by the local staff, according to directions supplied by us. From the data received the capacities were computed, employing the method described in the paper.

Curve	Pulp	Computed Capacity, Pounds per Square Foot	Actual Reported Capacity, Pounds per Square Foot	Number of Feet of Clear Solution Reported	Consistency of Feed Pulp	Consistency of Discharge Pulp
Fig. 4	Liberty Bell	4.87	5.94	1.25	10.00–1	2 00–1
5	Belmont....	14.1	14.75	1.5	7.00–1	2.11–1
6	Portland* (low-grade)	8.3	6.0	6 0	15.10–1	1.66–1
7	Nipissing (low-grade)	8.24	11.8	0 0	11.00–1	1.50–1
8	Presidio†...	33.0	17.65	6.0	5.62–1	1 58–1
9	Hollinger....	19.7	18.0	2.0	5.60–1	1.00–1
10	West End†.	15.24	11.97	5 to 6	6.10–1	2 02–1
11	Homestake.	7.78	7.05	....	33.00–1	2 18–1
12	Homestake.	8.86	8.55	....	17.50–1	1 50–1
13	Golden cycle (roasted ore)	19.3	19 1	0.0	7 70–1	1 00–1

Special Apparatus for Small-Scale Continuous Settling Tests

Although, as demonstrated, reasonably accurate results may be obtained by settling tests made in small glass cylinders (provided the results are properly interpreted), we recommend for testing a special device in which tests may be carried out under the same conditions of depth as would obtain in practice. Since the diameter of the cylinder, if over 2 in., does not in any way affect the settling rate, it is possible to construct from standard iron pipe and fittings, available in most localities, a device which will give results exactly parallel to those to be expected in practice. Fig. 14 shows in detail such a device, which in our experimental work has given very satisfactory results. The body or container is a piece of 4-in. black, iron gas-pipe, threaded at both ends (galvanized iron pipe or fittings should be avoided)on one end of which is screwed a 4-in. pipe flange, while on the other end is screwed a special cast-iron cone. Into the bottom of this cone is screwed a ½-in. nipple and connected to this is a ½-in. iron plug cock, into which is screwed another ½-in. nipple to prevent spattering. If it is not possible to procure the cone-shaped casting for the bottom, the same effect may be obtained by plugging the bottom of the pipe and screwing the discharge valve into a hole tapped at the center of the plug. A thin sheet-iron cone is

* *Portland.*—The bulk of this slime settles readily but there is a small amount of colloidal material which has a tendency to remain in suspension after the bulk of the pulp has settled. In order to obtain as clear solution as possible for precipitation, the thickener was operated below capacity, as indicated by the fact that there was 6 ft. of clear solution.

† *Presidio and West End.*—These thickeners were being operated very much below capacity, as indicated by the fact that there was 5 to 6 ft. of clear solution.

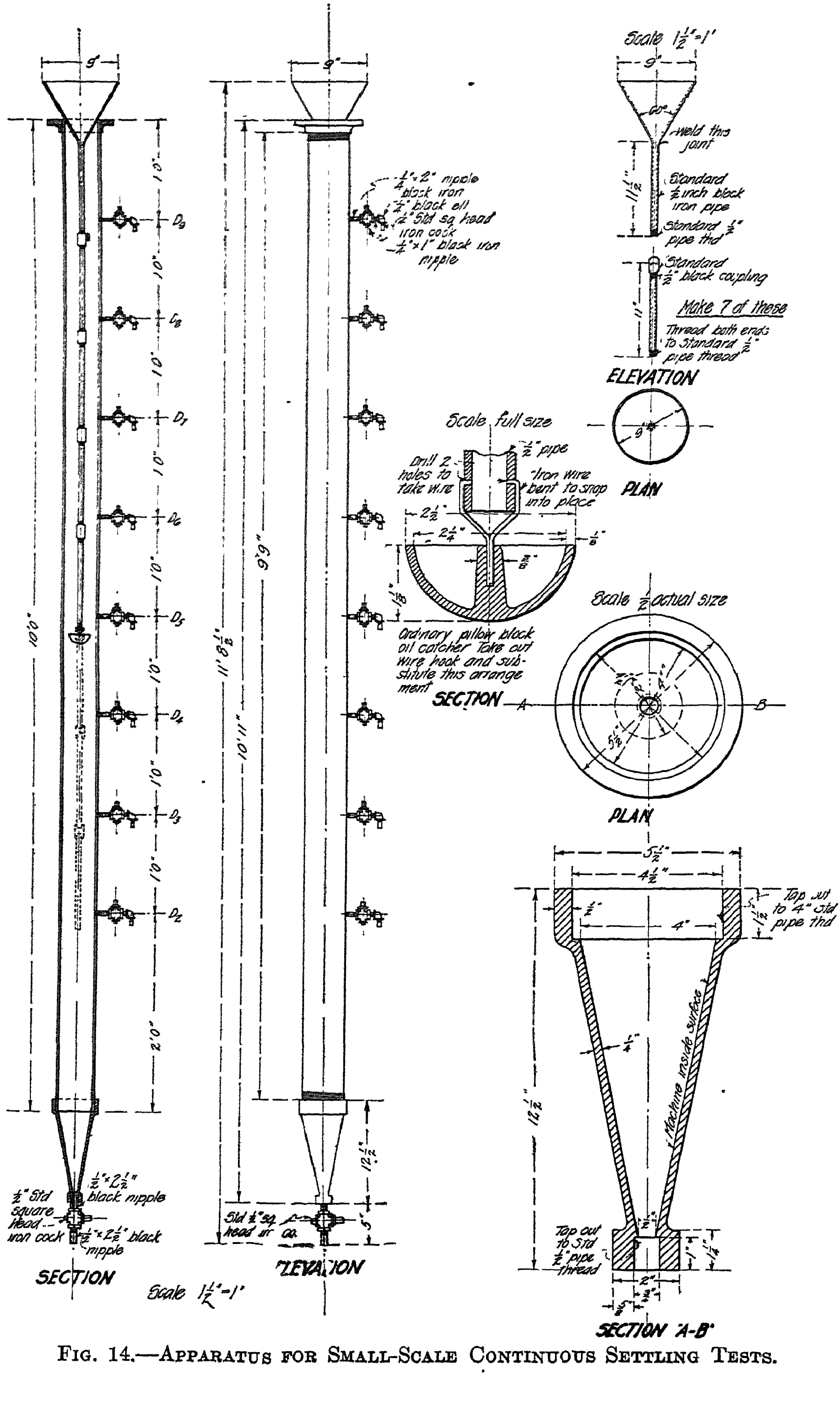


FIG. 14.—APPARATUS FOR SMALL-SCALE CONTINUOUS SETTLING TESTS.

made of such a size that the top just fits the pipe, while the apex is of the same size as the hole in the plug. This cone is slipped inside of the pipe and downward until it rests upon the bottom. Along the side of the 4-in. pipe, at regular intervals of 12 in., are bored holes tapped for a ½-in. thread. Into each of these is screwed a ½-in. nipple, to which is attached a ½-in. plug cock; to this, in each case, is attached an elbow and nipple to divert the outflow downward. The 4-in. pipe is arranged with a tripod or other suitable means of support so that it is maintained in a vertical position at a sufficient distance from the floor for the thickened pulp to be withdrawn readily from the plug cock at the bottom. Resting on top of the main pipe or container is a funnel for introducing the pulp. Attached to this funnel by means of iron sleeves are 11-in. lengths of black iron pipe, the idea being to make each section, with the sleeve when screwed into position, 12 in. in length and so arranged that the outlet from the stem of the funnel is somewhat below the particular cock on the side of the apparatus in use. Any depth of feed in multiples of 12 in. may be obtained by screwing on to the funnel tube the necessary number of 12-in. units. To the lowermost unit in use is attached a cup-shaped iron casting for the purpose of distributing the flow of pulp. This may be an ordinary 2½-in. cast-iron drip cup, such as is frequently used beneath bearings, attached in such a manner as not to obstruct the outlet of the pipe. The apparatus shown in Fig. 14 has a maximum available depth of 9 ft. for the thickening zone. Therefore, settling tests may be performed in steps of 1 ft., representing what may be expected in tanks where the thickening zone varies from 2 to 9 ft. in depth. If a study of the behavior of deeper tanks is desired, a longer containing pipe may be employed. To illustrate the use of this piece of apparatus: Assume that the settling behavior of a given pulp and the capacity per square foot of settling area of a tank having a thickening zone 5 ft. in depth are to be determined. First make a preliminary settling test in a glass cylinder according to the directions previously given. Let it be assumed that the results are as follows: The critical point in the glass cylinder is ascertained to occur at a dilution of 3 parts fluid to 1 part solid and the mean settling rate down to this point is 1 ft. per hour. After these data are obtained, proceed with the large test as follows: Through the feed funnel introduce pulp of a consistency of 3½ parts fluid to 1 part solids, until the large containing tube is filled to 5 ft. Since in a 5-ft. column of 3½ to 1 pulp, ½ part water is equivalent to 7.65 in., it follows that this particular pulp must settle a distance greater than 7.65 in. before it reaches the critical point. Therefore, 1 ft. of feed with a consistency of, say, 7 parts fluid to 1 part solids, may be added to bring the pulp surface up to cock *D*6. After 1 hr., test the settling by opening cock *D*5 to ascertain if clear fluid can be withdrawn. Close cock *D*5, open cock *D*6 and pour feed in through the funnel until a murky overflow

occurs at cock $D6$. Stop immediately the outflow at $D6$ by closing the cock. Allow another hour to elapse and test as before. Repeat this at hourly intervals until it is found that the pulp level has risen to cock $D5$, as determined by pulp flowing out at this cock. When this stage is reached, either the feed must be decreased or thickened pulp must be removed from the cock at the bottom. Next remove 6 in. of thickened pulp through the bottom cock. This pulp will be sandy and its specific gravity need not be determined. Feed may now be added at the original rate until pulp again appears at cock $D5$, when it becomes necessary to remove thickened pulp from the discharge cock. The amount of feed added should be ascertained by measuring the volume of both the overflow and discharge. Continue the process as described until the specific gravity of the discharge is uniform and the sand content normal. The feed pipe should extend slightly below the cock at the top of the zone tested and the surface of the pulp should never be permitted to fall below the mouth of the feed pipe, for, should it do so, it is not definitely known that the maximum feed is being given. If the pulp in the discharge is not as dense as desired, reduce the feed and continue the test. After equilibrium has been reached, the capacity per square foot per hour is computed. Further tests with various depths in the experimental apparatus will give the necessary data for choosing the depth of tank required. To the depth of thickening zone determined by the experimental apparatus must be added the additional necessary depth for storage, feed, etc., as previously explained. The series of tests for maximum capacity in the glass cylinder will greatly assist the operator in making the large tests.

Remarks Concerning Settling Tests

When we asked certain operators to make settling tests to prove the value of the formula $C = \frac{62.35R}{F - D}$ we were not familiar with the erratic behavior of certain thick pulps containing large proportions of lime, nor with the importance of small changes in temperature. We therefore limited the ratings to single readings on each test and did not mention the effect of temperature. It is probable, in the few cases where a fairly close check between actual and indicated capacity is not shown when the thickeners were being worked to the limit of their capacity, that the discrepancy is due to one or the other of these causes.

In our earlier work the pulp samples tested were built up to give the desired consistency by combining the discharge and feed, as previously described. Subsequently it was found more convenient to start with a sufficient measured quantity of feed pulp and use the same pulp throughout the whole series of tests for maximum capacity, hence each pulp sample

is now made up by decanting measured quantities of clear liquid and mixing the remaining pulp well before taking the sample for each test. A thorough mixture is very important, especially with tests which are to be settled for density. With thin pulps the settling rate will not be altered by the sand; for instance, if the sample has less than its proportion of coarse sand, the settling rate remains unaffected. The amount of electrolyte should be the same throughout all the tests and the same as that to be used in practice. A high proportion of lime almost invariably causes a somewhat thinner discharge from the thickener. In making tests for compression and final thickening in low cylinders, the clear overlying liquid should be removed in order to keep the surface of the liquid down near the pulp surface.

Before increasing the proportion of electrolyte in mill practice to increase the settling rate, the series of tests for maximum capacities should be made and carefully analyzed to ascertain if the increase in the electrolyte will prove advantageous in the zone of the pulp the settling rate of which is limiting the capacity of the tank.

The temperature of tests must be uniform and the same as will be maintained in practice. High temperature increases the settling rates in most pulps and it is not rare to find the rate altered from 30 per cent. to 50 per cent. by a few degrees change of temperature. The condition of the pulp tested is of utmost importance. Samples dried in the air or by fire may be ruined for testing. Many colloids when dried become set, that is, lose their plasticity. On the other hand, ore, as it comes from the mine, may not be in a condition to test. A fresh ore, ground to 200 mesh, settled to 40 per cent. moisture in an hour and required less than 2 sq. ft. of settling area per ton per day; after 5 hr. in contact with cyanide solution containing lime, it required 3 sq. ft. of settling area and settled to 50 per cent. moisture in 1 hr.; after 24 hr., it required 5 sq. ft. of settling area and 20 hr. to settle to 60 per cent. moisture; after 20 hr. no further change occurred. A sample of the same ore, ground wet and left in a sack for a week, required 10 sq. ft. of settling area and settled to 66 per cent. moisture after 24 hr. It is therefore obvious that pulp should have precisely the same treatment before and during testing that it will receive in practice.

It is not within the scope of this paper to discuss the action of electrolyte, but we desire to call attention to the fact that when the use of an electrolyte is desirable, frequently the most expensive electrolyte is cheaper to use than a less expensive one. Some of the electrolytes available for use in commercial practice are lime, caustic soda, alum, iron sulphate, permanganate of potash, sulphuric acid and calcium sulphate. At times the addition of a small proportion of a colloid may be more effective in aiding settling than the use of an electrolyte. We have had called to our attention a case in clarifying dirty water

where 0.02 lb. of glue per ton was much more effective than 1½ lb. of lime per ton.

Acknowledgment

We wish to acknowledge the facilities placed at our disposal for investigation in the field and the support given this work by the Nevada Mine Operators' association.

We are indebted to the following gentlemen who had their local staffs make the tests recorded in curves, Figs. 4 to 13, inclusive: Chas. A. Chase, of the Liberty Bell Mining Co.; A. H. Jones, of the Belmont Milling Co.; Thomas B. Crowe, of the Portland Gold Mining Co.; R. B. Watson, of the Nipissing Mining Co.; E. M. Gleim, of the Presidio Mining Co.; P. A. Robbins, of the Hollinger Gold Mine; Alvin Carpenter, of the West End Consolidated Co.; Allan J. Clark, of the Homestake Mining Co., and A. L. Blomfield, of the Golden Cycle Mining Co. We are also indebted to the Empire Co. and the Liberty Bell company for pulp samples, and to J. M. Tippett, L. B. Eames, and others, for making tests not recorded in the paper.

The Liberty Bell Methods of Precipitate Refining

BY A. J. WEINIG,* E. MET., TELLURIDE, COL.

(Arizona Meeting, September, 1916)

THE Liberty Bell cyanide precipitate is unique in that it is apt to vary widely in composition in the course of very short periods of time, and a method of refining and melting that would prove highly satisfactory at one cleanup would yield almost a hopeless "mess" at a second. Table I gives a general conception of the wide range in grade and composition of those raw precipitates. It is evident that no one standard flux could be expected to produce a high-grade bullion with such a widely variable precipitate, and that any method of treatment with acid requires most careful consideration.

Refining by Acids

Naturally, our first efforts to improve our precipitates were directed along the lines of acid treatment. Table II embodies the results of careful sampling and analysis of one cleanup at the time when wet refining reached its maximum development at this mill. We have now discarded all methods of wet refining, successfully replacing them by a more scientific method of flux calculation described herein, but it is believed that a general description of this wet refining is warranted as it throws some interesting side lights on acid refining in general.

TABLE I.—*Range of Composition of Raw Precipitates*

	Per Cent.
Gold and silver	25.0 to 75
Zinc	18.0 to 30
Lead	0.5 to 52
Copper	0.5 to 20
Silica	1.0 to 5
Calcium oxide	4.0 to 8
Sulphur	0.5 to 8

Column A gives the analysis of the raw precipitate; column B, that of the precipitate after heating with steam, with frequent addition of hydrochloric acid until a point was reached where the batch had a strong tend-

* Mill Superintendent, Liberty Bell Gold Mining Co.

ency to remain acid. The bath was then diluted with hot water, and repeatedly washed by decantation, until the liquor by test indicated that all soluble lime salts were removed. After the last decantation, commercial sulphuric acid was added, the bath heated by steam, and agitated with air until all action ceased. The bath was then diluted with hot water, and repeatedly washed by decantation, until by test all soluble zinc salts had been removed.

TABLE II.—*Composition of Products at Different Stages*

Substance	Raw Precipitate, Per Cent.	After Partial Treatment with Hydrochloric Acid, Per Cent.	After Hydrochloric and Sulphuric Acid Treatment, Per Cent.	After Treatment with Hydrochloric and Sulphuric Acids Followed by Caustic Soda, Per Cent.
Au and Ag	40.5	50.1	49.9	69.4
Zn	20.3	12.5	2.4	3.2
Pb	20.1	25.0	24.9	8 5
Cu	2.0	2.4	2.3	3.2
SiO_2	1.1	1.4	1.4	1.7
CaO	4.9	0 2	Trace	....
Total sulphur	1.81	1.84	5.53[1]	3.03[2]
Total	90.71	93.44	86.43	89.03
Final weight of precipitate derived from 100 lb. of raw precipitate, pounds.	100	80	81	59
Column	A	B	C	D

[1] 3.51 per cent. of this sulphur was combined as sulphate.
[2] 2.55 per cent. of this sulphur was combined as sulphate.

Column C gives the analysis of the precipitate after the sulphuric acid treatment. This analysis is unique. We note first a slight decrease in the grade of the precipitate in spite of the fact that a large portion of the zinc has been removed. The increase in total sulphur content is indicative, as it shows how well finely divided lead reacts with sulphuric acid to form lead sulphate. In this case a large proportion of lead sulphate was formed, thereby actually lowering the grade of the precipitate in spite of the removal of zinc. This explains why it often happens in treating with sulphuric acid raw precipitate containing much lime, as well as lead, that a precipitate is obtained actually lower in grade than the original raw precipitate.

Column D shows analysis of the precipitate after being treated by hydrochloric and sulphuric acids and lastly with hot caustic soda. To the residue, after decanting the last sulphuric acid wash, sufficient caustic soda was added to make a strong solution. The batch was then vigorously heated and agitated with steam for 2 hr., diluted with hot

water, and repeatedly washed by decantation, until, by test, all soluble lead salts had been removed.

The effect of the caustic soda bath is well shown by the final analysis. The grade was well brought up and a marked decrease in the quantity of lead and sulphur shows the solubility of lead sulphate in hot caustic soda.

Refining by Fusion

The present method of flux refining has been arrived at through a vast amount of experiment. In general, it embodies the principle of the addition of an oxidizer, and the proper proportionment of fluxes to carry off the oxidized impurities as metallic bases in a fluid slag.

A rather wide experience in the melting of precipitates has shown that the various impurities of precipitates are oxidized and driven into the slag in a definite order. Zinc tends to oxidize first, and the resulting zinc oxide will combine with the acid radical of the flux and therefore enter the slag. As in blast-furnace practice, the zinc tends to render slags infusible; and for this reason borax glass is always used in a greater amount than silica to furnish the acid radical for the slag. Borax glass greatly increases the fusibility of zinky slags.

Sulphur tends to oxidize second, and if an alkaline oxide is present in the melt, alkaline sulphate is formed, which is rather easily fusible. This sulphate is not soluble in the slag, and, being lighter, rises to the surface of the melt, forming the well-known sulphate "cover." The sulphate "cover" is nearly always quite pure, so that after any melt the melter may easily separate it from the slag, and, by weighing and computing its sulphur content, obtain a close check on the analysis of the precipitate. This cover is either Na_2SO_4 or K_2SO_4 depending on the alkaline oxide present. (The sulphate cover is an intensely interesting material to the chemist. Many unsuspected elements are known to concentrate therein. In the Liberty Bell material, Mo, As, Sb, Te, Se, and V have been detected.) Where precipitate is melted without oxidizers, sulphur generally makes itself known by the formation of matte—a very undesirable byproduct.

After all sulphur is oxidized, lead will oxidize, and the resulting lead oxide enters the slag proper. It is well known that lead oxide combines with silica in nearly all proportions under the influence of heat, and that the resulting lead silicates are easily fusible between rather wide limits. For this reason more silica may be used in the melt when much lead is present, and for the same reason the amount of borax glass may be diminished.

After lead, copper will oxidize to Cu_2O and combine with the acid radical of the slag. The color of such slags is red. Where an excessive

amount of oxidizer is present, copper will oxidize to CuO, and slag containing this oxide is green.

This selective oxidation is most clearly evident in the order of oxidation between copper and lead. It is practically impossible to drive copper into the slag as long as metallic lead is present in the bullion. Here quantitative separations between lead and copper are easily obtained.

Graphite crucibles have a marked reducing action on lead slags. No matter how rapid the melting, or how fast the resulting slag is removed from the crucible, if much lead is present, some will be reduced to metal. In such cases it is practically impossible, in direct melting, to slag copper, even though oxidizers are added far beyond the calculated requirement.

Where melts are conducted in clay or clay-lined graphite crucibles, this reducing action is of course avoided, and as a result copper will be found to enter the slag as easily as lead, though not until all lead is oxidized. Hence, in melting precipitate in graphite crucibles no attempt is made to slag copper. If a desirable grade of bullion can not be made unless copper is slagged, the melter will then, of course, use clay or clay-lined crucibles.

As an illustration of this, I give the data regarding a melt of precipitate containing a rather high amount of copper. This precipitate had received hydrochloric acid treatment only, and gave upon analysis the following composition:

	Per Cent.
Gold and silver	68.3
Zinc	8.7
Lead	8.3
Copper	9.5
Sulphur	1.1
Calcium oxide	0.1
Silica	0.9
Total	96.9

All of this precipitate having been fluxed alike with an oxidizing flux, equal portions were melted down in graphite and clay-lined crucibles. The resulting bullion from the clay-lined crucible shows 982 total fineness, while the bullion from the graphite crucible showed 812. I should add that, up to date, we have not found an entirely satisfactory clay or clay-lined crucible. At best they last one heat.

Fluxes.—The common oxidizing fluxes found on the market are manganese dioxide, potassium and sodium nitrate. Manganese dioxide can be obtained quite pure but at best it is costly because of its comparatively low quantity of available oxygen. It also requires extra weight of acid flux to retain the MnO in the slag. However, it gives

satisfactory results. Potassium nitrate is well known and needs but passing notice. Sodium nitrate is highly recommended and deserves special mention. It is very high in available oxygen and the crude product from Chile can be obtained practically pure so far as fluxing is concerned. It is laid down at Telluride at less than one-fifth the cost of potassium nitrate, and is one of the very cheapest fluxes on the market. It is also metallurgically more to be desired in the fluxing of sulphur, the resulting sodium sulphate being decidedly more fusible than the corresponding potassium salt and as a result the separations are better. The reaction between sodium nitrate and sulphur is $2NaNO_3 + S = Na_2SO_4 + 2\ NO$. Sodium carbonate or soda-ash is more desirable than the bicarbonate, since it goes considerably farther weight for weight as a flux, and lowers the freight bill. We use this flux only to supply alkali for the sulphur in the melt, when manganese dioxide is the oxidizer. Silica is an excellent acid flux, though its use must be guarded. Slags high in silica are not well adapted to the melting of precipitate, because of the higher temperature they require, especially in the presence of zinc. By reason of its cheapness, it is desirable to use as much silica as possible to replace borax in the melt, but our experience leads us never to go below a ratio of borax glass to silica in the slag of 2 to 1, where much zinc is present, or below a ratio of 1 to 1, even, where much lead is to be slagged.

In Table III, I have embodied results consistent with our practice and have arranged the same for convenience and rapidity of flux calculation.

Explanation of Table III.—In the substance column, I have placed the common impurities of the precipitate as well as the more frequently used fluxes. Column 1 contains the molecular weights of the substances in round numbers. Columns 2, 3 and 4, I have labelled "Per Cent. Oxygen." These columns indicate the weight of oxygen that the substance will combine with or evolve in per cent. of the original weight of substance. The terms "acid," and "base" oxygen I use for brevity. This is an effort to distinguish between oxygen that will enter the acid or basic part of the slag. The metallurgist's conception of a slag is that it is a fusible mixture of definite chemical combination between various acid and basic oxides. By "available oxygen" is meant the quantity of oxygen which the substance will evolve and which will be available in oxidizing metals or other impurities in the precipitate.

Column 5 gives the weight of manganese dioxide required to oxidize a unit weight of the substance on the same line in Column 1. These figures are derived from the chemical equation:

$$2MnO_2 = 2MnO + O_2$$

With pure manganese dioxide these figures represent approximately the amount of oxidation obtained in practice.

TABLE III.—*Flux Table*

Substance	Molecular Weight	Per Cent. Oxygen			1 Lb. Substance Requires, Pounds					
		Acid	Base	Available	MnO_2	KNO_3	$NaNO_3$	K_2CO_3	Na_2CO_3	
Cu″	63	....	25.4	12.7	1 40	1.07	0.91			
Cu′	63	..	12 7	.. ...	0 70	0.54	0 46			
Pb.	207	..	7.7	.. ...	0.43	0.33	0 28			
Zn	65	.	24.6	. ..	1.35	1.03	0.87			
Fe	56	.	28 5	.. ..	1.57	1.20	1.01			
S.	32	..	.....	.....	8 30	6 30	5 30	4.30	3.30	
SiO_2	60	53.4								
CaO	56	....	28.5							
MnO_2	88	..	18 2	18 2						
$NaNO_3$	85	...	9.4	28 2						
KNO_3	101	....	7.8	23.8						
Na_2CO_3	106	. .	15.1							
K_2CO_3	128	....	12 5							
Na_2O	62	. .	25.8							
K_2O	94	. ..	17.0							
$Na_2B_4O_7$	202	47.7	7.9							
Mixture of 2 parts borax glass and 1 part silica	... ..	49 6	5.3	. . .	..	.	....	..	...	39.0 per cent. available acid oxygen.
Mixture of 1 part borax glass and 1 part silica	....	51.1	3.2	..	..	.	....	....	. ..	44 7 per cent. available acid oxygen.
Mixture of 3 parts borax glass and 2 parts silica.	.....	50.0	4.8	....	..	..	....	...	..	40.4 per cent. available acid oxygen.
Column No	1	2	3	4	5	6	7	8	9	10

Columns 6 and 7 are similar to Column 5; they represent the weights of KNO_3 and $NaNO_3$ required to oxidize a unit weight of substance. The figures are derived from the equations:

$$2KNO_3 = K_2O + 2NO + 3O$$

and

$$2NaNO_3 = Na_2O + 2NO + 3O$$

Authorities seem to agree that these equations represent the breaking up of the niters in presence of an oxidizable substance with heat; and I find that this agrees well with practice, where the melts are conducted in graphite crucibles. It has been frequently noted, however, that, when the melts are conducted in clay or clay-lined crucibles, the oxidizing power of the niters is markedly higher than is indicated by these equations; the reason being that graphite has a reducing action on slags. When clay or clay-lined crucibles are used, about 80 per cent. of the niter called for by the above equations will give the same oxidation in practice.

Columns 8 and 9 give the weight in pounds of potassium carbonate or sodium carbonate required to form potassium sulphate or sodium

sulphate with 1 lb. of sulphur which has been oxidized to SO_3. Thus $K_2CO_3 + SO_3 = K_2SO_4 + CO_2$. We use these carbonates only when manganese dioxide is the sole oxidizer.

Column 10 indicates the quantity of available "acid" oxygen in mixtures of borax and silica. This table could be carried out infinitely but I have only listed mixtures that experience and practice indicate to be feasible.

In "Flux Calculation, Case I," given in the appendix, I have gone through the regular flux calculation with some detail. For the type precipitate I have taken the lowest grade of raw precipitate that we have yet melted.

The analysis of this precipitate was:

	Per Cent.
Au and Ag	25.0
Pb	38.8
Zn	20.5
Cu	1.4
S	1.2
CaO	2.0
SiO_2	1.4
Total	90.3

It will be noticed that I use a type slag of 2 of acid oxygen to 1 of basic oxygen. This is the result of experience. Such slags have just a noticeable tendency to string and are the best, most easily fusible, and least corrosive to the crucible. They are invariably of low grade, and have been made repeatedly as low as $20 per ton in precious metal. This type of slag has been found by far the most desirable, and we now make it as a regular practice, irrespective of the composition of the precipitate or fluxes used.

Since 100 lb. of precipitate was assumed at the start, 92 per cent. of this flux was added to the estimated dry weight of precipitate. It is interesting to note that this melt gave a resulting bullion fineness of 972 total fine. As we assumed that all the copper would go into the bullion, and that the balance of the impurities and base metals were removed in the slag we did expect a bullion of $25 \div (25 + 1.4) \times 1{,}000 = 950$ total fine, approximately. This melt, of course, was conducted in a graphite crucible.

"Flux Calculation, Case No. 2" was the calculation for the precipitate treated with caustic soda, listed in Column D, Table II. The bullion made was 942 total fine, which is in remarkable accord with the calculated fineness, assuming that all copper went into the bullion, as $69.4 \div (69.4 + 3.2) \times 1{,}000 = 955$ total fine.

The calculation shows the use of soda ash and manganese dioxide.

It is also interesting that a distinction was made between sulphide and sulphate sulphur.

Sintering.—After removing the precipitate from the press the computed flux is added and the whole well mixed while wet. The mixture is then loaded into cast-iron pans, and placed in a large cast-iron muffle furnace; and the temperature of the furnace is gradually raised to a red heat. In this operation moisture is removed, and the precipitate gradually agglomerates and settles to about one-third of its original volume. During this process practically all the chemical reactions are completed and the mass sinters to a point where subsequent "dusting" is avoided.

When sintered the charge is taken from the furnace and cooled, and is easily removed from the pans. This gives an excellent product for crucible melting. In one case, where a precipitate containing 90 per cent. of gold and silver was obtained by acid-treatment, the melting of over 11,000 oz. of fine bullion was done in 4 hr. in one No. 275 oil-burning tilting-furnace. This included the time taken in starting up the furnace in the cold. With precipitate of average grade, *i.e.*, 65 per cent. Au and Ag, a melting rate of 1,300 oz. of bullion per hour is common.

The advantages gained by sintering are many. The completion of practically all chemical reactions in the muffle furnace prevents any excessive boiling in the crucible, and the crucible may be loaded to the top during the period of melting. The slow application of heat in the muffle furnace gives ample chance for complete chemical reaction. Hence, we arrive, in the use of oxidizers, at practical results which correspond with theory. The great decrease in the volume of the precipitate during sintering enables us to get much more weight in the crucible at a charge, and this again decidedly increases the melting rate. Metal losses during the sintering are exceedingly small. At one time the vents of our muffles were connected to a dust chamber, but nothing was ever recovered; and frequent tests of the gases leaving the muffles, by passing them through water or cotton soaked in a solution of sodium sulphide, failed to show more than traces of valuable metal.

These flux calculations have now been used for a period of 18 months, during which time about 60 melts have been conducted on all grades of raw and acid-treated precipitate; and in no case has there been a failure to realize a close approximation to the grade of bullion calculated.

Slags.—The slags vary so widely in composition that at a glance no relationship seems to exist. They all, however, conform to one single relationship: namely, the ratio of "acid" to "base" oxygen is approximately 2:1.

After reviewing the analyses of some 20 slags, I find that the range of the composition of slags may be considered to fall within the following

limits. The slags were made from all grades of raw and acid-treated precipitate.

	Per Cent.
Zn	5 0 to 30
Pb	0.5 to 65
Cu	0.0 to 20
CaO	0.0 to 10
SiO_2	8.0 to 20
Mn	0.0 to 25
*B_2O_3	10.0 to 20
*Na_2O	0.0 to 18
*K_2O	0.0 to 20

The composition of the sulphate cover remains quite constant.

When soda niter or sodium carbonate are used the sulphur in the sulphate cover is from 20.5 to 21.5 per cent. This is close to the theoretical requirement of sulphur in Na_2SO_4. When potassium niter is used the sulphur content lies between 17 and 18 per cent., likewise close to the theoretical requirement for sulphur in K_2SO_4.

As an illustration of the distribution of the bases and compounds in all the products of the melt, I will take Case 1. Here 134 lb. of slag and 5½ lb. of cover were made by actual weights for every 100 lb. of dry precipitate melted.

A partial analysis of this slag gave:

	Per Cent
Zn	15.4
Pb	28.0
Cu	0.2
CaO	1.7
SiO_2	14.1

The sulphur in the cover was not determined. If we therefore compute the actual weights of materials in precipitate and flux and in the slag, we arrive at the following figures:

Precipitate and Flux 100 Lb. Precipitate		Pounds	Slag 134 Lb.	Pounds
Zn $0.2050 \times 100 =$		20.5	Zn $0.154 \times 134 =$	20.60
Pb $0.3880 \times 100 =$		38.8	Pb $0.280 \times 134 =$	37.50
Cu $0.0140 \times 100 =$		1.4	Cu $0.002 \times 134 =$	0.27
CaO $0.0200 \times 100 =$		2.0	CaO $0.017 \times 134 =$	2.30
SiO_2 $0.0140 \times 100 =$	1.4		SiO_2 $0.141 \times 134 =$	19.00
In flux	18.0			
SiO_2 Total		19.4		

If we assume that the sulphate cover contains an average of 21 per cent. sulphur, the 5.5 lb. of cover would indicate $5.5 \times 0.21 = 1.16$ lb. sulphur or 1.16 per cent. of dry precipitate. This is a very close check to the 1.2 per cent. obtained by actual analysis; and we thus arrive at

* Estimated from weights of borax and alkalies used.

results that are entirely satisfactory within the limits of experimental error.

Slag Balance			Oxygen	
Substance	Weight, Pounds	Combined as	Acid, Pounds	Base, Pounds
Zn	20.60	ZnO	....	5 10
Pb	37.50	PbO	. ....	2.90
Cu	0.27	Cu_2O	...	0 03
CaO	2.30	CaO	. .	0 66
SiO_2	19.00	SiO_2	10.2	....
*B_2O_3	27.00	B_2O_3	18.5	....
*Na_2O	22.50	Na_2O	... ...	5.80
			28.7	14.49

* Computed from weights of borax and niter used.

Totals in the original flux calculation as 28.89 : 14.50 or a ratio of approximately 2 to 1.

The cyanide slags are stored, and once a year are melted up in a small blast furnace with brick charges, made from the ash obtained by burning old mill launders and tanks. This furnace which has a capacity of 5 tons of slag and brick a day also handles copper scrap directly, and will deliver about 15 tons of pig copper per day. This local smelting lasts only about 10 days a year. Any matte made is controlled by running the furnace more oxidizingly and thus driving the metals into the bullion and slag. As a result of this the only byproducts shipped are lead bullion and pig copper.

In spite of the seemingly high costs of local smelting, the ultimate gain is greater, by reason of the invariably greater recovery, as compared with that realized by shipping the material away.

Appendix

Flux Calculation Case No. 1

Data and Explanatory (Calculations by slide rule)	Pounds: Oxygen, Acid	Oxygen, Base	$NaNO_3$	Borax Glass	Silica
(References apply to Table III) assume 100 lb. of precipitate					
Zinc 20 5 lb.					
20 5 × 0 87 (See Col. 7, opposite Zn) =	.....	.....	17 90		
20.5 × 0 246 (See col. 3, opposite Zn) =	...	5.04			
Lead 38 8 lb.					
38 8 × 0 28 (See Col. 7, opposite Pb) =	...	....	10.90		
38.8 × 0.077 (See Col. 3, opposite Pb) =	..	3.00			
CaO 2 0 lb.					
2.0 × 0.285 (See Col. 3 opposite CaO) =	....	0 57			
SiO_2 1.4 lb.					
1.4 × 0.534 (See Col. 2, opposite SiO_2) =	0.75				
Borax Glass (as it is desirable to maintain a ratio of borax to silica of 2 to 1 in the slag it is convenient to add here the proportionate amount of borax to balance silica in the precipitate. In this case the silica is low and this correction could readily be omitted on this particular precipitate but as it is necessary to correct for borax here when silica is high the correction for silica is carried through for form)					
Hence 2 × 1.4 = 2.80 lb. borax	.....	.	.....	2.80	
2.80 × 0.079 (See Col. 3, opposite $Na_2B_4O_7$) =	.....	0.22			
2.80 × 0.477 (See Col. 2, opposite $Na_2B_4O_7$) =	1.34				
The total pounds of $NaNO_3$ added is	.....	.....	(28 80)		
As this places extra basic oxygen in the slag as Na_2O we have:					
28.80 × 0.094 (See Col. 3, opposite $NaNO_3$) =	....	2.71			
Total acid and basic oxygen	(2 09)	(11.54)			
The 2 09 lb. acid oxygen will balance ½(2.09) = 1.05 lb. base oxygen. This leaves 11.54 − 1.05 = 10 50 lb. base oxygen to be satisfied, or 21.0 lb. extra acid oxygen to be added. Using the mixture of 2 borax and 1 silica to supply this acid oxygen, we have 21 ÷ 0.39 (See Col. 10, opposite Mix. 2 B.G. to 1 SiO_2) or 54 lb. of mixture. This is:					
36 lb. borax glass	......	.....	...	36.0	
18 lb. silica	....	...	....	.....	18
also					
54 × 0 496 (See Col. 2, opposite Mix. 2 B G. to 1 SiO_2) =	26.80				
54 × 0.053 (See Col. 3, opposite Mix. 2 B.G. to 1 SiO_2) =	....	2.86			
Grand totals of acid and basic oxygen or a ratio of 2 to 1 approx.	28.89	14 50			
Sulphur—As all sulphur goes into the "cover" as well as the ni er used to oxidize it, we have: 1.2 × 5.30 (See Col 7, opposite) =	...	...	6.37		
Grand totals of fluxes to be used	....	...	35.17	38.80	18.0

The melter made up the flux by parts as:

	Parts
$NaNO_3$	35
Borax	39
Silica	18
Total	92

Flux Calculation Case No. 2

Data and Explanatory (Calculations by slide rule)	Pounds: Oxygen: Acid	Oxygen: Base	Na_2CO_3	MnO_2	Borax Glass	SiO_2
(References apply to Table III) assume 100 lb of precipitate						
Zinc 3.2 lb.						
3.2 × 0 87 (See Col 7, opposite Zn) =				2.80		
3 2 × 0 246 (See Col. 3, opposite Zn) =		0.79				
Lead 8 5 lb.						
8.5 × 0 43 (See Col. 5, opposite Pb) =				3.65		
8.5 × 0.077 (See Col. 3, opposite Pb) =		0 65				
Silica 1 7 lb.						
1.7 × 0.534 (See Col. 2, opposite SiO_2).	0 89					
Borax 2 55 lb.						
As the slag will be low in zinc we choose a lower ratio in the slag of borax to silica for economy. Taking the ratio of 3 of borax to 2 of silica we must here take into account the necessary borax to balance the silica in the precipitate.						
Hence: 1.7 × ½ × 3 = 2.55 lb. borax					2.55	
2.55 × 0 079 (See Col. 3, opposite $Na_2B_4O_7$) =		0.21				
2 55 × 0.477 (See Col. 2, opposite $Na_2B_4O_7$) =	1.22					
Manganese dioxide to oxidize sulphide sulphur						
3 03 − 2 55 = 0.48 lb. sulphide sulphur						
0 48 × 8 30 (See Col. 5, opposite S) =				4.00		
Hence we have 10.40 lb. MnO_2 required to oxidize metals and sulphur				(10.45)		
Therefore the base oxygen which the MnO_2 carries into the slag is 10.4 × 0.182 (See Col. 3, opposite MnO_2) =		1.90				
Totals of acid and base oxygen	(2.11)	(3.55)				
2.11 lb. acid oxygen will satisfy ½ (2.11) or 1.06 lb. base oxygen. This leaves (3.55 − 1.06) = 2.49 lb. base oxygen to be satisfied						
$\frac{2 \times 2\ 49.}{0.404}$ (See Col. 10, opposite Mix. of 3 borax to 2 silica)						
Equals 12.3 lb. of the acid mixture.						
This is 7.4 lb. borax					7.40	
and 4.9 lb. silica						4.90
Also 12 3 × 0.048 (See Col. 3, opposite Mix. 3 to 2) =		0.59				
12.3 × 0.500 (See Col. 2, opposite Mix. 3 to 2) =	6.15					
Grand totals of acid and base oxygen or a ratio of 2 to 1 approx.	8 26	4.14				
Soda ash to form Na_2SO_4 with all the sulphur forms "cover" and is not considered a portion of the slag.						
3 03 × 3.30 (See Col. 9, opposite S) =			10.00			
Grand totals of fluxes to be used			10.00	10 45	9.95	4.90

The melter therefore made up the following flux:

	Parts
Na_2CO_3	10
MnO_2	11
Borax glass	10
Silica	5
Total	36

Mining and Milling Practice at Santa Gertrudis

BY HUGH ROSE,* PACHUCA, MEXICO

(Arizona Meeting, September, 1916)

THE properties of the company lie within the Pachuca district, State of Hidalgo, Mexico, connected by three railway lines with Mexico City, 55 miles southwest, and by two lines with Vera Cruz, 250 miles southeast.

The ores were formerly divided by sorting into two classes, smelting and milling, the former averaging about 2 oz. gold and 335 oz. silver, the latter 0.12 oz. gold and 23 oz. silver. The smelting ores were sold to custom plants, principally the American Smelting & Refining Co. at Aguascalientes. The milling ores were treated by the patio process at the Guadalupe Hacienda, at Pachuca. This patio was probably the largest in existence at the time and continued in active operation up to March, 1910.

In January, 1910, the mines and patio were sold to English interests, represented by Camp Bird Limited. Two new companies were formed, the Compañia de Santa Gertrudis, S. A., to operate the mines, and the Compañia Beneficiadora de Pachuca, S. A., to build and operate a custom cyanide milling plant. (Fig. 1.)

Geology

The historical geology of the Pachuca district is described by Fred J. Pope as follows:

1. The most recent sedimentary rocks in the district belong to the Cretaceous period.

2. The Cretaceous rocks were penetrated by large intrusions of andesite (referred to hereafter as primary andesite) which occur as elliptical dome-shaped masses, striking northwest-southeast. These andesite bodies are from 8 to 15 miles long and from 6 to 10 miles wide.

3. A period of erosion during which the contour of the andesite domes was changed.

4. Andesite flows (secondary andesite) which rest unconformably on the primary andesites.

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* General Manager, Compañia de Santa Gertrudis, S. A.

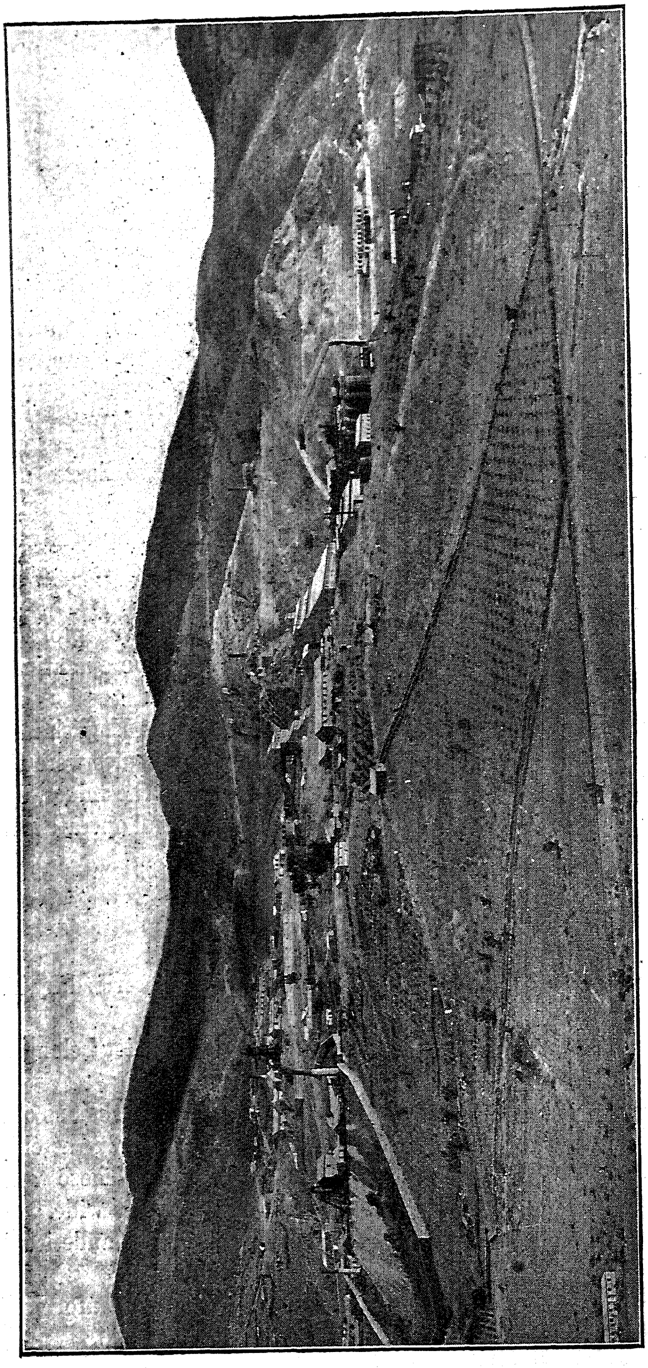

FIG. 1.—SANTA GERTRUDIS MINE AND MILL.

5. A second period of erosion, again exposing areas of primary andesites.

6. A series of rhyolite flows which partially covered the primary and secondary andesites.

7. A third period of erosion which exposed large areas of both the primary and the secondary andesite. During this period there were many intrusions of quartz-porphyry dikes which cut the primary and secondary andesites and the rhyolite. For the most part they do not affect the economic tenor of the veins. In occasional instances, however, the veins are locally impoverished where they cut these dikes. Relatively to the extent of the orebodies, these dikes are very small and practically have no economic bearing.

8. A period of areal faulting, probably due to subsidence and adjustment caused by the large, dome-shaped masses of primary andesite cooling at depth. During this period fissures were formed, rock between the walls was brecciated and there was a heavy silicification, but practically no economic values were brought into the veins. It was essentially a period of fissuring, brecciation and silicification.

9. A second period of faulting with movements relatively much less than during the first period. It was during and immediately subsequent to this secondary faulting that the veins received their economic values.

10. Erosion and minor faults.

Country Rock

In discussing the magmatic variations of the primary andesite, Mr. Pope further states.

The primary andesite in which the Santa Gertrudis vein occurs is not a homogeneous rock. There are local variations due to differentiation in cooling. For the most part the andesite has a slight grayish-green color, is porphyritic, and breaks with a rough surface. This is the prevailing normal type and averages approximately 65 per cent. silica. A more basic variation averages a little over 60 per cent. silica, is porphyritic, dark green in color, breaks with a conchoidal fracture and is very tough. Between these two types there are all gradations and one often graduates imperceptibly into the other. Both occur at all levels to the bottom of the mine, a depth of 2,000 ft.

Operations in the Pachuca district have demonstrated that the tough basic phase of the primary andesite is not favorable to the formation of ore and that all the large orebodies are found in the normal type. This is due to the fact that after the primary faulting and silicification, the basic phase was as tough or tougher than the original rock and thus resisted

the secondary fracturing and subsequent introduction of economic values.[1]

Nature and Formation of the Veins

The veins are not simple fissures filled with silica and economic minerals, but in a general way may be described as crushed zones varying from a few feet up to 50 ft. and in some instances over 100 ft. in width. Usually both foot and hanging walls are fairly well defined, but in many places the dividing line between ore and wall rock can be determined only by systematic sampling.

As a rule the hanging wall is heavy, due to fracturing and to slips or faults parallel to the vein, and in consequence must be carefully supported when mining the ore.

Ore Occurrence

The veins of the district are notable for the size and persistence of the ore shoots.

As a rule, the ore shoots do not outcrop but are discovered at varying depths of one hundred to several hundred feet from the surface, and in the upper levels are shorter and often have a distinctive pitch, generally to the east. Usually, several of these shoots combine in depth to form the main great ore shoot of the mine. In the Santa Gertrudis vein a number of the smaller shoots were thus worked to a depth of from 800 to 900 ft. At about this horizon they joined together, forming an orebody which persisted for another 600 to 700 ft. in depth, having a length in the several adjoining properties of about 3,000 ft. and a width averaging 18 ft. The grade of this great ore shoot was not less than $20 per ton.[2]

At a horizon of about 1,800 ft. from the surface, the veins, though still strong, become rapidly impoverished, and at 2,000 ft. little payable ore is to be found. Below this horizon explorations are being conducted to discover the possible existence of a new zone of mineralization, but such work is not sufficiently advanced to permit any forecast of results.

The ores generally are clean, carrying very small quantities of lead, zinc or iron, and little or no manganese, all of which are unimportant economically. The principal economic mineral is silver sulphide—argentite. The commercial values are confined to gold and silver, which are

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[1] Mr. Pope's report was made after a brief examination undertaken at the time of the purchase of the properties by the Camp Bird interests. Subsequent developments over 5 years have confirmed his findings in general. Several ore-bearing veins occur in basic andesite areas but ore shoots are small and of relatively small economic importance. In the main North Vein, wherein the principal ore shoots are located, the lean parts of the vein within the ore-bearing horizon usually occur where the vein passes through the tough basic phase.

[2] U. S. currency and dry short tons used throughout unless otherwise specified.

present in the proportion of 15 per cent. and 85 per cent. respectively. By weight, for each 1,000 oz. of silver there are usually 5 oz. of gold present.

Mining Systems and Methods

The mining systems in general use are:

(*a*) Overhead stoping without timbering, with filling.

(*b*) Overhead stoping with timbering and filling.

(*c*) Overhead stoping with shrinkage; with and without subsequent filling.

In mining virgin ore in place, system (*a*) is most generally adopted, being used where the hanging wall is too heavy to employ shrinkage stoping, but sufficiently strong to hold until the stope can be filled. Fig. 2 illustrates the details of this system. Foot-wall drifts (counter-drives) in the harder andesite are kept open after the sill floor of the vein has been stoped out and filled. Crosscuts from the counter drive are run through the vein every 165 ft. and heavily timbered with chutes for drawing the ore from the stopes. These chutes are carried up with cribbed timbers as the stope advances and are divided into two compartments, one for ore and one for manway. Between each two crosscuts a raise is put up to provide entrance for the waste used in filling. Spreading of this waste is done with wheelbarrows.

Throughout the Pachuca district the re-mining of the old areas has become of great importance. The old fills are generally payable and in most cases ore in place is also found on the walls. At first sight it might appear that such territory would be cheaper to mine than virgin ground. But this is not the case, for both the hanging wall and the ore are heavier, making it necessary to adopt system (*b*) with resulting slower extraction of ore at higher cost. Fig. 3 illustrates the details of this system which in the main follow those of system (*a*) with the addition of square-set timbering. As the timbers are soon covered by the waste filling, light sets of 8 in. square material or of equivalent round posts are sufficient. Sets measure 7 ft. 5 in. high by 5 ft. by 5 ft. in the clear.

System (*c*) is applicable only in one small stope in the Santa Gertrudis mine, and hence need not be discussed in detail. In general it is used more commonly in the Real del Monte district than on the Pachuca side, the walls in the former district being much harder and more secure.

Levels were formerly driven 75 to 100 ft. apart. More recently the distance has been increased to 165 ft. (50 m.) vertically. It is questionable whether a greater distance than this could be used economically because of difficulty of holding open the timbered chutes.

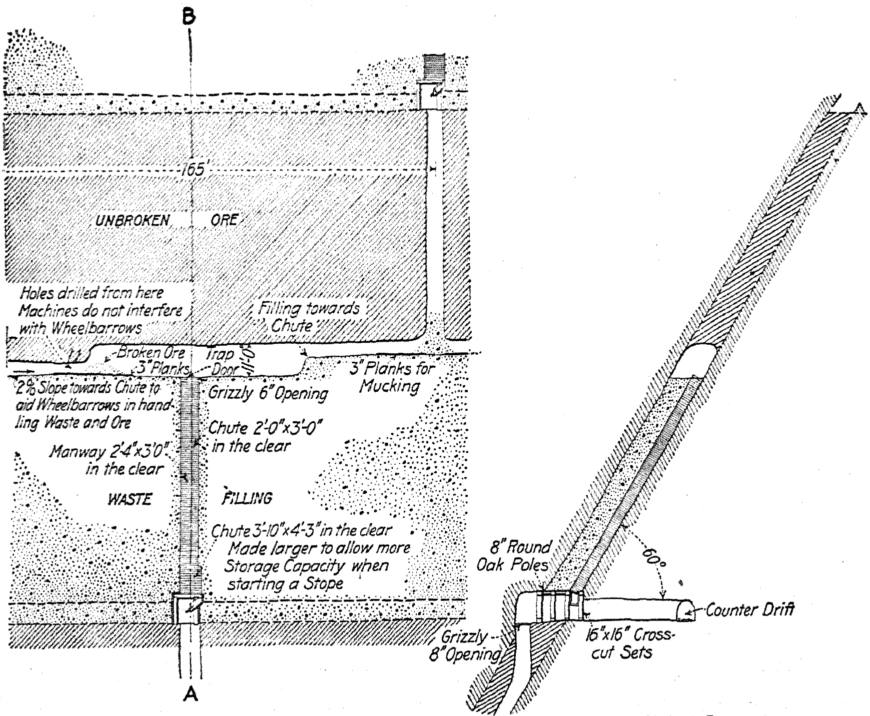


FIG. 2.—OVERHAND STOPING WITHOUT TIMBERING AND WITH FILLING.

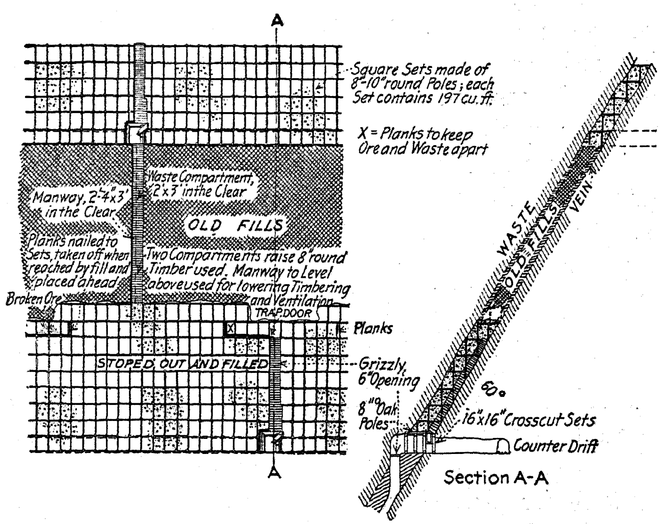


FIG. 3.—OVERHAND STOPING WITH TIMBERING AND FILLING.

Ore Breaking

Ore breaking is entirely overhand, using air-hammer drills and ⅞ in. 0 per cent. gelatine powder. Filling for the stopes formerly obtained rom caving stations in the hanging wall, is now derived partly from levelopment headings. To provide excess requirements a quarry is eing opened on the surface. A central raise in hard foot-wall rock is eing put up to tap this supply. This raise will deliver or receive waste t each level.

Development

Shaft sinking is generally done against considerable water, 200 to 00 gal. per minute; 3⅛-in. machine drills are used. Electric firing of ounds has been tried, but with only partial success, due to the breaking imbers which must be carried close to the bottom. Electric time uses have been used with better success and when intelligently handled re an improvement upon ordinary fuse firing.

Of special interest was the construction of a two-compartment shaft San Francisco shaft No. 2) adjacent to the existing three-compartment an Francisco shaft No. 1. A pillar of rock, 20 ft. wide, was left to proect the latter shaft, which was kept in operation; sinking and raising vere carried on coincidently from several different levels. The shaft aises were timbered in two compartments, a light cage being used in the nanway to handle men and tools. A small electric hoist at the foot of he raise served the cage, the hoisting rope being carried over a sheave wung from a movable beam at the top of the raise timbers. Sinking vas done by usual methods.

Work was commenced in March, 1912, and the shaft completed o a depth of 1,768 ft. from the collar to the 19th level, with pockets at number of levels, and put into operation in May, 1913. Steel timberng, bought cut to proper lengths but fabricated locally, was installed hroughout. No. 18 galvanized corrugated sheet lagging was used, but nly where necessary, being slipped into place on the outside of sets and eld secure by small steel clips, riveted to the sheets and fitted over the lange of the I-beam wall plates. The cost of the shaft without pockets vas:

	Per Foot
Raising and sinking	$24.62
Timbering (steel)	20.33
Loading and tramming	0.65
Hoisting (waste to surface)	6.23
Explosives	2.92
Lighting	0.59
Miscellaneous	0.27
	$55.61

All drifting and crosscutting of importance is done with 3⅛-in. machine drills, using the horizontal bar mounting, two machines to a bar, the muckers cleaning up the breakage while the top part of the round is being drilled.

Raising is entirely accomplished with air-hammer drills.

Winzing is generally by hand and only a relatively small amount of it is done, raising being generally cheaper and more rapid. Air-hammer drills are occasionally used in winzing in hard ground.

Average Development Costs per Foot

	Shaft Sinking (against Water)	Drifting and Crosscutting	Raising	Winzing
Labor	$59.25	$3.75	$2.50	$4.75
Timber	8.45	0.10	0.35	0.10
Explosives	3 25	1.70	0.70	0.60
Lighting	0.70	0.05	0.06	0.06
Tracking and piping	1.00	0.60	0.12	
Power	15 60	0.50	0.50	0.55
Maintenance, machine drills	0.50	0.05	0.10	
Tools	0.45	0.10	0.10	0.10
Tool sharpening	1.50	0.60	0.60	0.10
Hoses, pump fittings, etc.	1.50	0.05	0.05	
Lubricants	0.25	0.02	0.02	
Supervision	1.05	0.30	0.30	0.30
Totals	$93.50	$7.82	$5.40	$6.56

Tramming

Tramming underground is by hand and electric haulage. Four main levels are equipped with electric haulage using 2.5-ton Jeffrey 250-volt trolley locomotives. A battery locomotive was tried without success, the equipment being insufficient to handle the load against varying grades and track curvatures. In the locomotive haulage, a train is made up of 10 U-dump cars each of 20 cu. ft. capacity, equal to about 0.9 dry ton of Santa Gertrudis broken ore. The cars are equipped with Whitney roller-bearing wheels and, in general, are giving good satisfaction. The center of gravity is sufficiently low to require complete filling for easy dumping and to allow the cars to be emptied over the shaft pockets without stopping the train. Waste, when required, for stope filling, is loaded into the empty cars from the waste chutes on the return trip. On hand-tramming levels, the ore is passed, wherever possible, to electric-haulage levels for transportation to shafts. Tracks are 20 in. gage, with 16- and 20-lb. rails.

Hoisting

Men and material are handled in San Francisco shaft No. 1, by two double-deck cages operating in and out of balance. These are served by an Allis Chalmers double-reduction geared electric hoist, using a three-phase, 50-cycle, 440-volt, 300-hp. induction motor with grid type resistance, at a rope speed of 1,000 ft. per minute; also in San Juan shaft, two single-deck cages operating in balance and served by a converted 150-hp. electric hoist, with a rope speed of 700 ft. per minute.

Ore and waste are handled in San Francisco shaft No. 2 by two 5-ton skips operating in or out of balance. These are served by a Fraser and Chalmers single-reduction, herringbone-geared hoist using a three-phase, 50-cycle, 440-volt, 350-hp. induction motor with liquid controller type resistance, at a rope speed of 1,000 ft. per minute, and a capacity of 800 tons per 8-hr. shift; also in San Guillermo shaft, one 3.5-ton skip operating counterbalanced and served by a duplicate of the hoist used on San Francisco No. 1, the rope speed being 1,000 ft. per minute and the capacity 300 tons per 8-hr. shift. This compound shaft is of interest because of its unusual characteristics. It is of old-style masonry from collar to 9th level. It is heavily timbered through the vein to the 13th level and has a twist of 30° between the 13th and 15th levels to bring it parallel to the vein. Between the 15th and 16th levels a 50-ft. radius compound converts the shaft to 60° incline at which it continues to the 20th level, a total vertical and inclined length of 2,090 ft. Except in the heavy part through the vein, little trouble has been experienced in the operation of this shaft, despite its small size in the upper section.

Fig. 4 shows details of construction of San Francisco No. 2 skip, of special interest being the trunnion wheels, the duplicates of which have given no trouble in use in San Guillermo compound shaft. These wheels are also utilized in dumping the skip, as shown in Fig. 5. The loop in the dumping path engages the bottom wheel, thereby preventing the skip from falling back into the shaft in case the safety detaching hook should not hold during an overwind. This dumping path is designed to take care of a skip for vertical or inclined service.

Fig. 6 shows electric-hoist brake-release contact brought into play by being struck by the ascending bale of the skip cage. The hoist brakes, air-operated, are under control of a solenoid connected to the brake weights. When the electric contact is made, the solenoid drops these weights independently of the engineer, thereby applying the brakes.

Plough-steel ropes of flattened-strand construction are used, 1 in. in diameter on men and materials hoists and 1⅛ in. diameter on the ore hoists. Ropes are systematically inspected, following South African requirements. After 2 years' service, new ropes are put on, even though the old ropes show no undue wear.

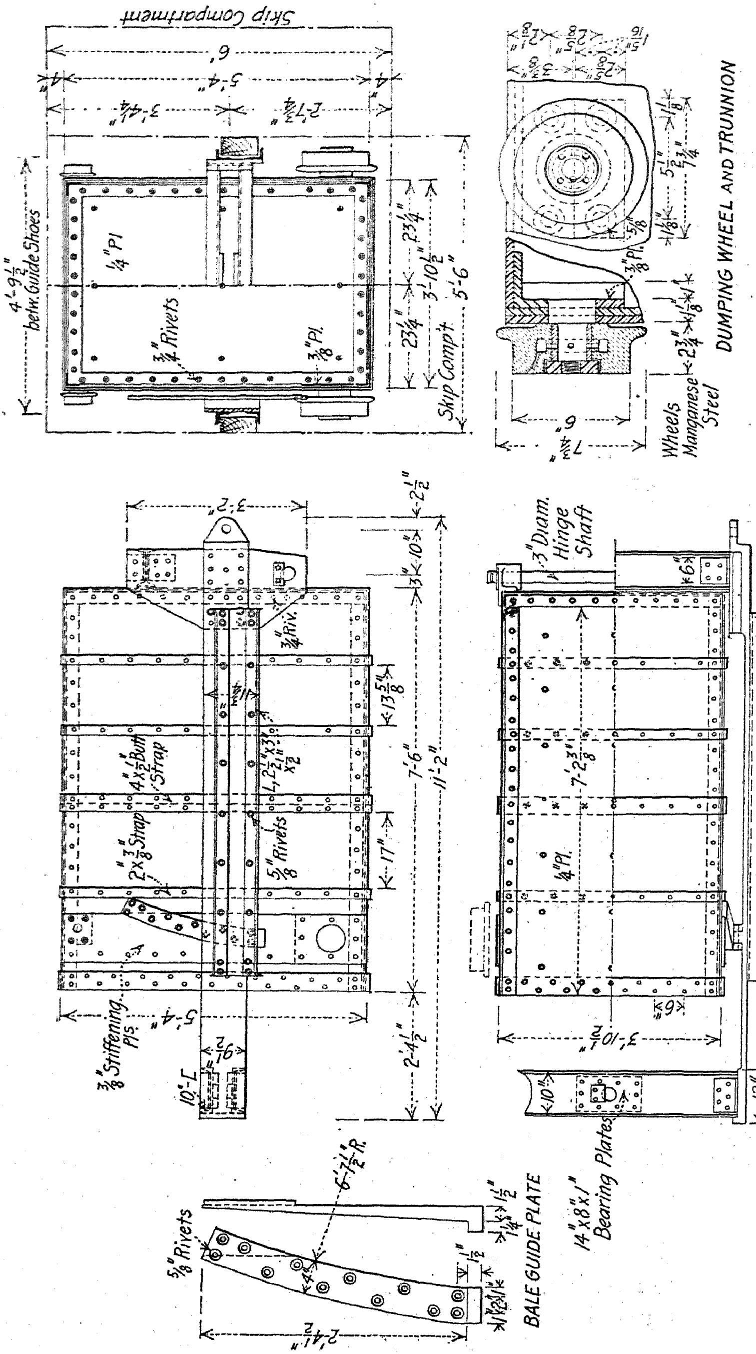

Skip Compartment
betw. Guide Shoes
¼" Pl
¾" Rivets
⅜" Pl.
Skip Compt
DUMPING WHEEL AND TRUNNION
Wheels Manganese Steel
¾" Riv.
4x½" Butt Strap
2x⅜" Strap
⅝" Rivets
⅜" Stiffening Pls.
3" Diam. Hinge Shaft
¼" Pl.
14"x8"x1" Bearing Plates
BALE GUIDE PLATE
⅝" Rivets

FIG. 4.—SKIP.

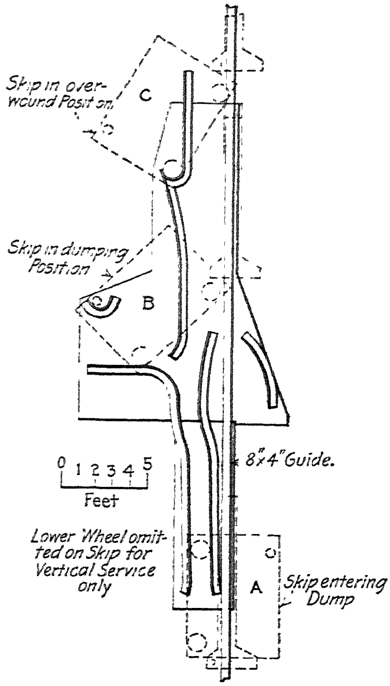


FIG. 5.—DUMPING PATH IN HEADFRAME.

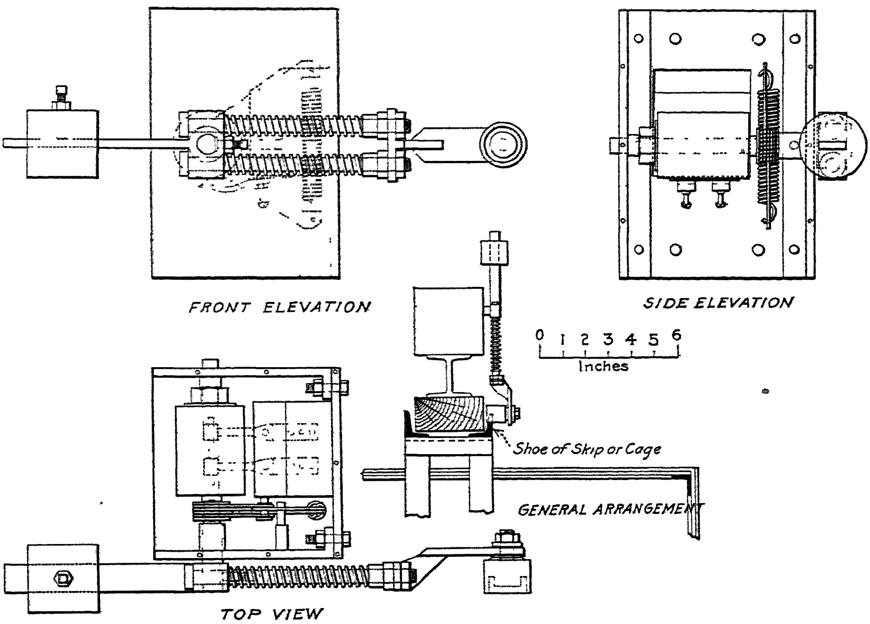


FIG. 6.—ELECTRIC-HOIST BRAKE-RELEASE CONTACT.

Pockets serving the skips are located on all working levels. Fig. 7 shows the characteristic arrangement of the grizzly on top of the pocket and the tracking layout; Fig. 8 shows the standard loading station in general use. Because the ore is free-running and unusually fine, no

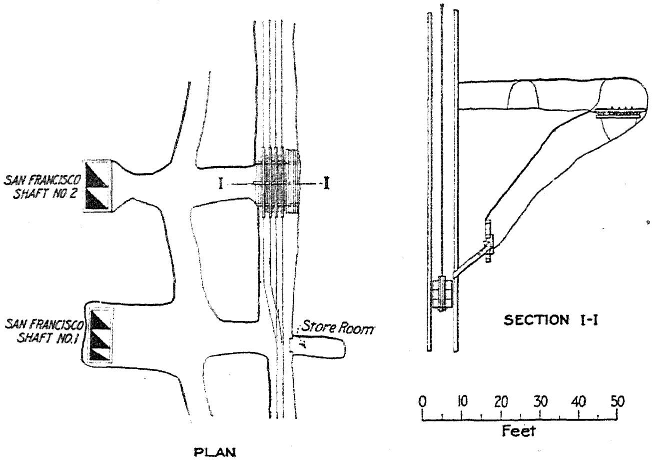


FIG. 7.—STATION AND POCKET LAYOUT.

measuring chutes are required, skip loading being accomplished accurately and with minimum loss of time by manipulation of the gates. With ample ore supply, *i.e.*, pocket fairly full, the loading time is from 15 to 20 sec.

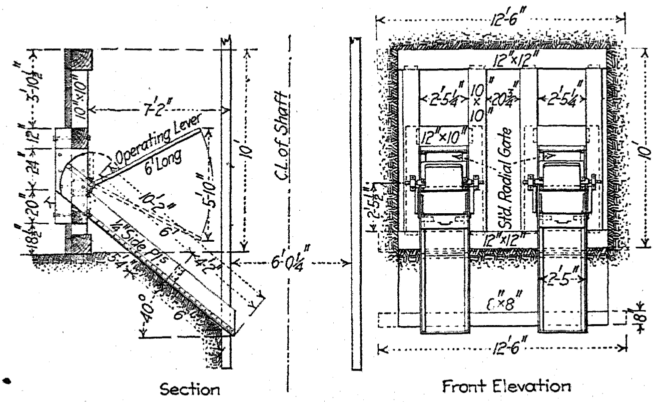


FIG. 8.—SHAFT-LOADING STATION.

Signals from cage or skip tender to hoist engineer are by electric bells, the same system being used by persons calling at any level. Bells are Holtzer Cabot polarized loud-ringing type, 300 ohms resistance, operating on 50- or 60-cycle, 110-volt alternating current. Current is taken from a 1,000-volt power circuit and transformed to 110 volts. The

three-wire system is used for the bells of each shaft compartment. Where telephones are also installed, both bell and telephone wires are in one wire-wrapped, lead-covered cable containing several reserve wires for emergency use. No. 18 size wires are used. Fig. 9 shows the waterproof pull switch used in signaling, which has given satisfactory service.

Mining Drainage

The main pump station is located on the 18th level near the San Guillermo shaft. The 10-in. discharge column, of Mannesman drawn-steel tubing with bolted forged-steel flanges and round copper gaskets, is carried horizontally on concrete pillars to the San Francisco shaft No. 1 and thence vertically to the tunnel level, a total length of 2,067

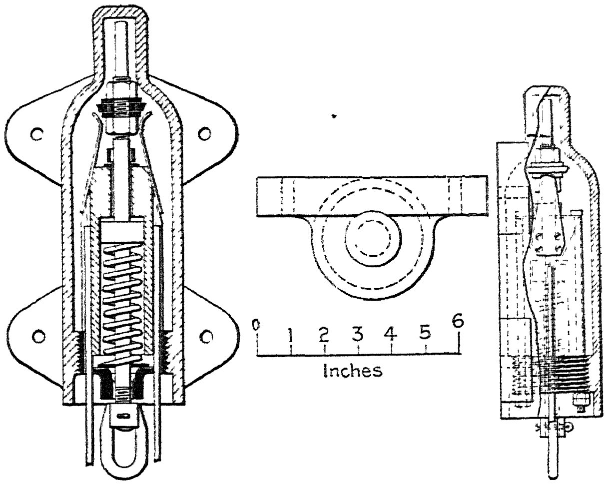


FIG. 9.—SIGNAL SWITCH.

ft. and a vertical lift of 1,328 ft. The equipment of the station consists of three 500 gal. per minute, horizontal, duplex, double-acting Prescott high-duty pumps, each driven through one reduction of herringbone steel gearing by a three-phase, 50-cycle, 1,040-volt, 250-hp. General Electric induction motor, operating at 480 r.p.m.

No water-hammer difficulties have developed during 3 years' service and the equipment in general has given excellent satisfaction. A traveling crane made the installation easy and assists materially in repair work. The pumps were installed under a guarantee of 85 per cent. efficiency based on power delivered to motor pinion. The motor efficiency was measured at the factory. A series of tests made by the local staff, after the pumps had been in operation for over a year, gave the results shown in Table 1.

The old main pump station was at the 15th level and consisted of two eight-stage Sulzer centrifugal pumps each direct connected to 480-

hp. induction motors of German make, no-load speed, 1,200 r.p.m., capacity 3,600 liters per minute each. These motors were designed for 60-cycle current and at 50 cycles failed to handle the water satisfactorily because of reduction in speed. One of these units is now installed on the 18th level and, being connected to the suction of the pump remaining on the 15th level, the two now serve as a reserve unit of about 1,000 gal. per minute capacity.

Water from the 20th level is pumped to the 18th level by a 1,000 gal. per minute Alberger centrifugal pump, electrically driven. In shaft

TABLE 1.—*Results of Test on Pumping Equipment*

No. of Pump	Water Pumped, Gallons per Minute	Static Head, Feet	Static Head, Pounds	Dynamic Head, Feet	Dynamic Head, Pounds	Theoretical Water Horsepower	Input to Motor, Horsepower	Motor Efficiency from Curve	Input to Pump, Horsepower	Pump Efficiency, Sump to Discharge	Combined Pump and Motor Efficiency
1	480	1,328	573	1,371	591	160	193	93.1	180	89.2	83.0
2	481	1,328	573	1,371	591	161	199	93.1	186	86 6	80.7
3	469	1,328	573	1,371	591	157	192	93.1	179	88.9	81.6
1 & 2	950	1,328	573	1,388	599	317	408	93.1	380	83 6	77.8
1 & 3	940	1,328	573	1,388	599	314	400	93.1	372	84.4	78.5
2 & 3	939	1,328	573	1,388	599	314	403	93.1	375	83.6	77.8
1, 2 & 3	1,421	1,328	573	1,418	612	475	650	93.1	605	78.4	73.0

Motor data, 480 r.p.m., 250 hp., 1,040 volts.

Pump data, 49.6 r.p.m.; length of stroke, 3 ft.; original diameter of plunger, 4⅝ in.; diameter of plungers at time of test, 4⁹⁄₁₆ in.; calculated displacement of plungers at 4⅝ in. diameter, 519.4 gal. per minute; at 4⁹⁄₁₆ in. diameter, 505.5 gal. per minute.

Temperature of water, 87° F.

Weight of water, 8.304 lb. per U. S. gallon.

NOTE.—After the tests a small stick was found under one valve in pump 3.

Average gallons per minute pumped, 815; per day, 1,173,600; average pumping cost, including auxiliaries per day, $208.24; per gallon, 0.0177 c.

sinking below the 20th level, No. 10 Cameron sinking pumps are used with air at 80 lb. pressure, and as the lift becomes too great for the air pumps, electric centrifugal station relay pumps are installed.

Ventilation

The natural ventilation of the mine is good, due to the many shaft connections as shown on the longitudinal projection (Fig. 10)—and to communications with eastern adjoining properties, Barron and La Blanca. A motor-driven Sirocco fan is occasionally used temporarily to ventilate some section pending the completion of connections for natural air currents.

Sanitation

Particular attention is paid to keeping all parts of the mine free from refuse or filth. Special sanitary buckets are provided in suitable places

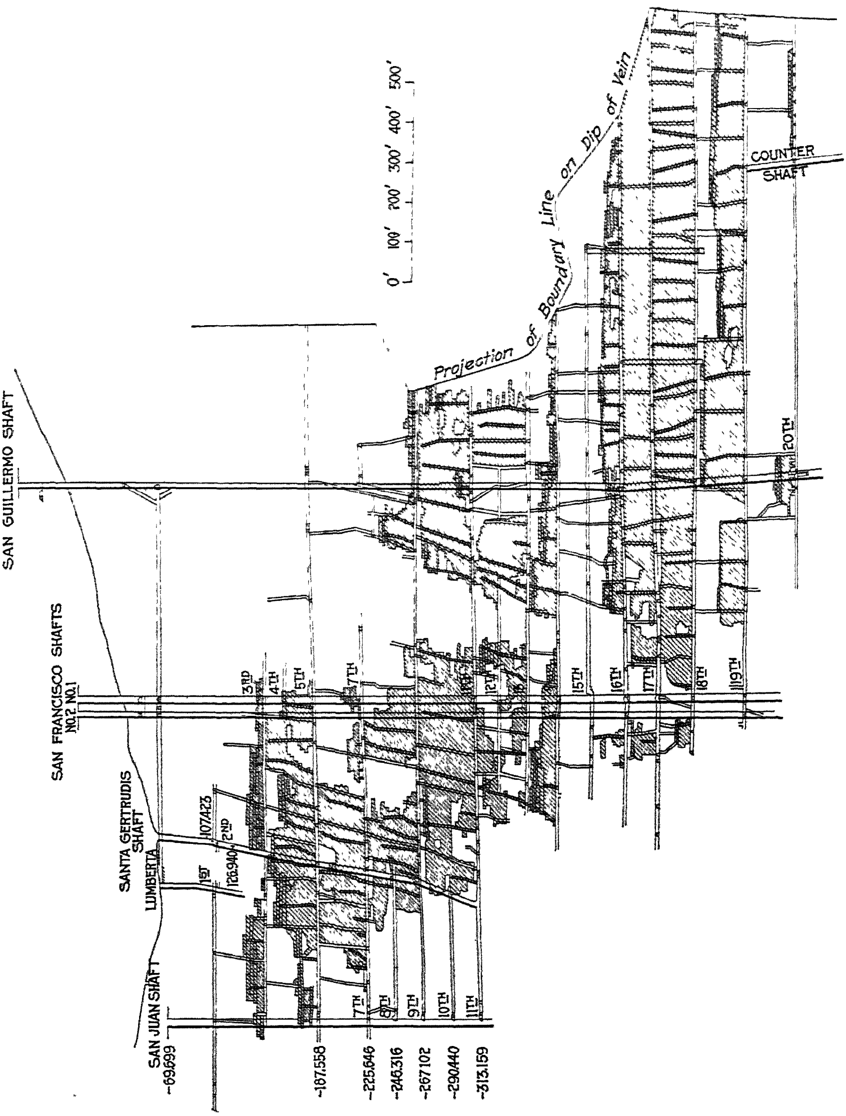


FIG. 10.—VERTICAL PROJECTION. NORTH VEIN.

on each level which the men are required to use and which are systematically taken to the surface for cleansing, lime being applied liberally underground to limit odors.

The main levels, where there is much traffic, are sprinkled with water daily, using a small truck fitted with a barrel having an ordinary sprinkler attachment.

Good drinking water is piped to each shaft station.

A well-ventilated and heated change room is provided, equipped with shower baths of hot and cold water. This facility is not ordinarily found at a mine in Mexico, but its use is to be encouraged.

Safety-First Measures

Within the last two years, an active safety-first campaign has been in hand with very gratifying results. As is well known to all operators in Mexico, the native labor is unusually careless of danger or ordinary precautions against accident. Hence the problem presents exceptional difficulties and requires all the more careful attention to preventive details. Because of the reckless irresponsibility of the individual, coupled with a callousness to accidents which in fairness must be charged to the operators as well as to the men, the accident rate in Mexican mines is abnormally high. And for the reasons stated, this rate will always compare unfavorably with those of well-regulated mines in the United States or Europe. Nevertheless, a great deal can be done to improve this condition.

After securing information from some of the leading American mining companies, experienced in safety-first measures, similar steps were undertaken here. A book of instruction covering accident prevention and first-aid measures was issued in Spanish and English. The shift bosses or "capitanes" of each mine section were made directly responsible for accidents, careful statistics being kept of the work of each boss, with a monthly premium of money to the one having the best record for the month, and a gold watch with suitable engraved inscription for best record for 6 months. An annual contest is held in which various crews compete in rescue and first-aid measures. Bosses and men are drilled in the use of oxygen helmets. A monthly bulletin is printed in Spanish and distributed among the men, in line with similar bulletins issued by American companies.

As of interest to other operators in Mexico, some of the mechanical measures undertaken will be described briefly. All shaft entrances are provided with self-closing gates. After trying out several types, the one selected as most simple and efficient is as illustrated by Fig. 11. The gate is merely swung off center as shown and closes by gravity without undue jar. Fig. 11 also shows the upper gate similarly swung, to protect the individual from falling rocks while standing at the shaft.

Many accidents in Mexican mines are from men carelessly stepping into open chutes or manways in stopes or levels. All chuteways through-

out the mine are now covered with rail grizzlies; manways either have a trapdoor of light corrugated iron or, where this interferes with ventilation, the timber is carried up sufficiently so that the top can be covered, leaving an opening at one end or on the footside for passageway. This opening is kept small enough so that a man cannot slip into it inadvertently. All manway openings on levels have, in addition, a warning red electric light.

Explosives to be used currently are stored underground in suitably located crosscuts, fitted with steel doors set in masonry. Fuse and caps are similarly stored separately. It having been found impossible to

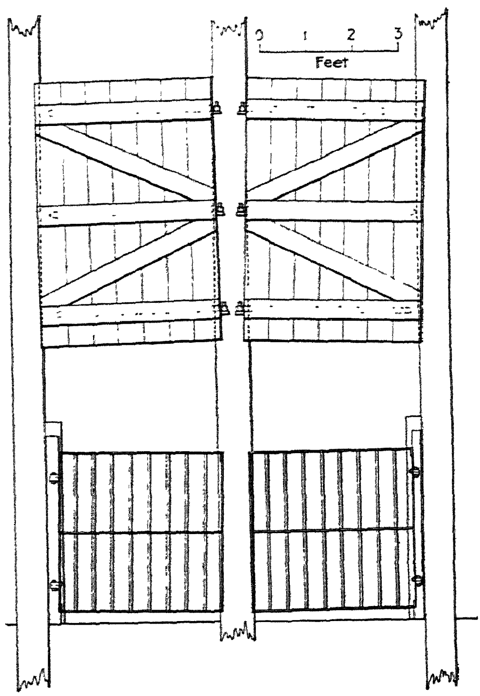


FIG. 11.—SELF-CLOSING GATE USED AT SHAFT ENTRANCES.

prevent the practice of crimping the caps with the teeth, fuse is now issued from these underground stores, cut to desired length and with caps properly crimped on by the storekeeper. The main storage of explosives is located some distance from the property.

Rock falls are a principal cause of many accidents. As elsewhere, the main preventive is constant vigilance on the part of the bosses combined with such coöperation as it is possible to secure from the men. This spirit of coöperation is obtainable only in a limited amount in Mexico, but continued application will bring it forth to a helpful extent.

Following are accident statistics per 10,000 shifts, before and after safety-first measures were undertaken:

Accidents

	Deaths	Requiring Operation	Trivial	Total
Before	0 574	1.436	12.067	14.077
After	0.334	0.669	8.358	9 361

NOTE.—Trivial accidents include numerous bumps, scratches, slight cuts, etc., which do not incapacitate men from working.

MINING COSTS

For convenience of comparison and of preserving complete records in compact form, the monthly costs are entered upon a tracing form from which blueprint copies are taken. The system requires perhaps less work than where similarly detailed costs are typewritten, whereas the advantage is obvious over any method which does not permit continuous column comparison of items.

Average costs are given below. These do not include recent results which, because of special influences, have been too low to present fairly what should be obtainable under normal conditions:

	Per Dry Short Ton
Development	$0.52
Ore breaking	1.47
Mine to mine bins	0.24
Drainage	0.24
Surface expense	0.04
General expense	0.27
Total mining cost	$2.78

These results are on the basis of a daily production of 1,000 short tons of ore and may be taken as fairly representative of the Pachuca district. In the adjoining Real del Monte district, lower costs are obtainable because the harder walls permit the adoption of the shrinkage system of stoping with consequent reduction, if not elimination, of expenditure for square-set timbering and filling. At Santa Gertrudis, for each ton of ore extracted, two-thirds of a ton of waste must be supplied and handled for filling, and 17 ft. board measure of timber used. The shrinkage system affords another important advantage in Mexico by providing a reserve of broken ore which may be drawn upon more heavily at times to make up for current deficiencies due to numerous feast days and a relatively low constancy of labor. Another important

factor affecting local mining costs is the light weight of ore in place and filling ore (old fills). The former averages 13.6 cu. ft. per ton; the latter averages 17.2 cu. ft. per ton. At Santa Gertrudis, the moisture in the ore averages 5.0 per cent., and filling ores represent 65 per cent. of the total tonnage mined.

A fair grade of timber is obtainable at $20 per 1,000 ft. board measure. Other supplies, if carefully purchased in quantities, cost about the same as in a mining district in Colorado or Nevada.

Labor efficiency is low, the complicated mining system likewise adversely affecting the duty per man. Including all labor in stopes for drilling, timbering, filling, etc., 1.27 tons of ore per man-shift are broken and placed in the chutes; based upon all underground labor, including bosses and foreman, 0.687 ton of ore per man-shift is delivered to the surface.

Power is purchased at from $45 to $50 per horsepower-year from the Mexican Light & Power Co. whose hydro-electric plant is about 75 miles away.

Surface Transportation

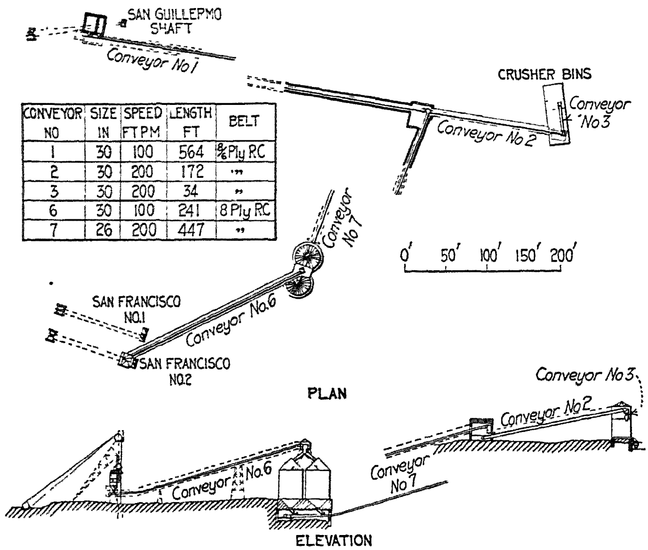
All ore is delivered from the mine by self-dumping skips, discharging into surface ore bins. Thence it is transported to the crushing-plant ore bins by means of belt conveyors. Figs. 12 and 13 show in plan the conveying system from the San Francisco No. 2 and San Guillermo shafts to the crusher bins, and also a cross-section of the line from San Francisco shaft No. 2.

Because of the previously mentioned free-running quality of the ore, plus the unusual percentage of fines which reach the belts first and form a cushion or bed for the larger rocks, the system operates with minimum wear or trouble. Hand-operated sector gates are used throughout, the chute lips being hinged and counterweighted to swing up from the belt when not in use.

On account of re-mining old fills, a large amount of decayed wood comes up with the ore and is picked off at the belts. Careful sorting tests have proved that waste picking cannot be done at a profit. Under close supervision, 3.5 per cent. by weight was picked off, averaging $0.70 per ton at a cost of $0.30 per ton of waste; increasing the waste sorted out to 5 per cent. resulted in an increase of value to $1 at a cost of $0.40 per ton of waste. The actual cost of passing these grades of material through the mill, less value recovered, is calculated to be lower than the sorting cost plus recoverable value lost. Therefore, waste sorting has not been adopted. Sorting high-grade ore for shipment to smelter, because of small quantity available as against high milling efficiency, is not profitable.

This "Mine Bins to Mill Bins" cost has averaged $0.02 per dry short

ton. Belt wear is exceedingly low. Belt No. 2 (Fig. 12) is the only one which has been replaced once against a total of 1,131,708 tons handled to date. Belt No. 1, originally installed and put into operation in June, 1911, has handled 691,999 tons of ore and from all appearances is good for a repetition of this duty. Belts 6 and 7, a later installation, have been in operation since May, 1913, handling a total of 439,709 tons and showing little wear.



CONVEYOR NO	SIZE IN	SPEED FT PM	LENGTH FT	BELT
1	30	100	564	8/6 Ply RC
2	30	200	172	"
3	30	200	34	"
6	30	100	241	8 Ply RC
7	26	200	447	"

FIG. 12.—PLAN CONVEYING SYSTEM.
FIG. 13.—SECTION VIEW.

MILLING

The milling business is conducted by a separate organization on a custom basis. The capacity of the plant is 1,100 tons per day.

Before taking up details for construction and operation, a condensed description of the mill flow scheme (Fig. 14) will be given.

The crushing-plant ore bins of 600 tons capacity feed two No. 6K gyratory crushers, discharging to a double screen 14 ft. long. The oversize from the 4-in. round-hole screen passes by conveyor to the tube-mill rock-storage bin, while the undersize joins the oversize of the 2-in. round-hole screen for crushing in two No. 4K gyratory crushers. The undersize from the 2-in. screen, together with the discharge from the secondary crushers, is delivered to a 22-in. troughed-belt conveyor, equipped with a Merrick weightometer.

The 22-in. conveyor discharges over the first one of three Vezin samplers, a 5 per cent. cut being taken and stored in a sample bin of 15

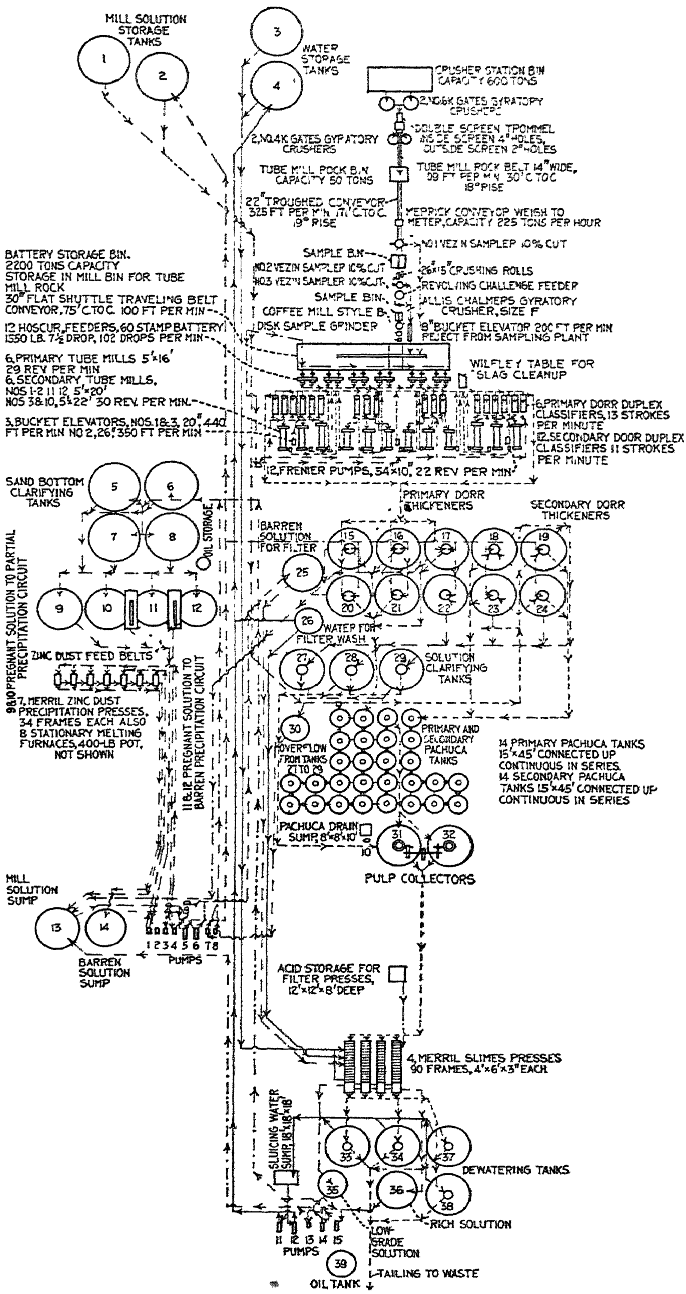


FIG 14.—MILL FLOW SHEET.

tons capacity. The reject flows to a 30-in. shuttle-type flat-belt conveyor for distribution into battery bins of 2,000 tons capacity. The first sample cut is fed from the 15-ton bin through a set of 26 by 15-in. rolls reducing to ¾ in. and finer, and discharging to a second Vezin sampler cutting out 10 per cent. which is delivered by a revolving Challenge feeder to the third Vezin taking a 20 per cent. cut. The sample thus obtained is 1 ton for each 1,000 tons milled and is crushed to ¼ in. in a size F gyratory crusher, thereafter being cut down by Jones riffles and reduced in the usual manner. Rejects from the second and third Vezin samplers and from the quartering floor are returned to the battery bins by an 8-in. elevator.

The ore passes from the battery bins through Hoscur feeders to 60 1,550-lb. stamps arranged in units of 10 stamps each, 20 stamps being driven by a 65-hp. motor, belted to a jack shaft; the stamps make 102 7½-in. drops per minute; 3-mesh and 4-mesh screens are used, the pulp from the batteries flowing through split distributing launders to six primary duplex Dorr classifiers, the sand passing to six 5 by 16-ft. tube mills, the discharges from which are delivered to the launder leading to the secondary classifiers by elevator or by reserve 10 by 54-in. Frenier pumps, one pump to each mill. The slime overflow from the primary classifiers is laundered to eight secondary duplex Dorr classifiers, which feed the sand to four 5 by 20-ft. and two 5 by 22-ft. tube mills, the discharges from which are returned by elevator or Frenier pumps to the classifiers.

The slime overflow from the secondary classifiers passes to six primary Dorr thickening tanks, 35 ft. diameter by 15 ft. deep, wherein the pulp is thickened from 10 to 1 to 1.5 to 1.

The thickened pulp is delivered to a set of 14 primary Pachuca tanks, 15 by 45 ft., operating in series; the discharge from the last tank in the series is by means of an air lift, submerged in the tank itself, and delivering to a launder where a wash of four parts of mill solution is applied, the pulp flowing to four secondary Dorr thickening tanks, 35 by 15 ft., wherein it is again thickened to about 1.75 to 1 and delivered to a set of 14 secondary Pachuca tanks, 15 by 45 ft., also operating in series, the discharge from the last tank of the series flowing to two 35 by 12-ft. storage tanks equipped with mechanical agitators.

These storage tanks feed by gravity at 35 lb. pressure four 90-frame Merrill filter presses, size of leaf 4 ft. by 6 ft., width of frame 3 in. The tailing sluiced from these presses flows to four 35 by 15-ft. Dorr thickening tanks for recovery of water before passing to the tailing storage dams.

The pregnant solution is clarified by passing through four sand filter tanks, 40 ft. diameter by 10 ft. deep.

Precipitation is effected by the Merrill zinc-dust process in two

circuits, partial and barren, in order to economize in zinc dust. Two presses are in use on the barren circuit and three on the partial.

The precipitate is melted in a battery of eight oil-fired No. 400 crucible furnaces arranged on the arc of a circle and served by a radial jib crane, fitted with an air cylinder for raising and lowering the pots.

All pumping of solution and water is concentrated in two pump houses, one located below the filter plant and the other below the precipitation plant. The pumps are of vertical triplex plunger types, gear-driven by motors, and are installed so that one pump is in reserve for two circuits.

An electrically operated inclined tramway runs from the top to the bottom of the mill delivering material to any floor.

Details of Construction and Operation

Compactness of design was sacrificed in order to secure a gravity flow. Such design was permissible owing to an unusually ample mill site of 17° slope, coupled with a mild climate requiring no housing of tanks. Supervision is made easy because the size of the plants warrants the division into two departments, mill and cyanide, and the tramway is used by the bosses in getting round.

The mill is electrically driven throughout, 50-cycle alternating current being distributed at 440 volts.

Crushing, Weighing and Sampling

The crushing-plant design exhibits no points of special interest. The elimination of elevator returns and the favorable character of the ore make the operation unusually easy and simple. Consumption of steel liners, etc., is almost negligible. The 6K gyratory crushers, driven by 30-hp. motors, take only 15 hp. each. This is again due to the character of the ore and the large percentage of fines which, if the ore had greater abrasive quality, it might pay to screen out beforehand.

An ample supply of mine rock, over 4 in. in size, for use in the tube mills in place of pebbles, is obtained cheaply from the revolving-screen oversize as described above.

The Merrick weightometer is checked weekly against a weighed quantity of ore, the average error being well under 1 per cent. Current weighings are corrected by the weekly factor thus obtained.

Particular attention is paid to securing an accurate sample of lthe ore delivered to the mill. Ordinarily two classes of ore are samped separately each day. The rejects from these two are thoroughly mixed and quartered down, to make the mixture sample for the day. The calculated average assay of the two class samples checks very closely the

assay results of the mixture. The usual samples are taken for moisture, which averages about 5.0 per cent.

Stamping and Tube Milling

The design of the stamp battery follows standard lines except in a few minor details. The mortar-box foundation bolts are crossed as shown in

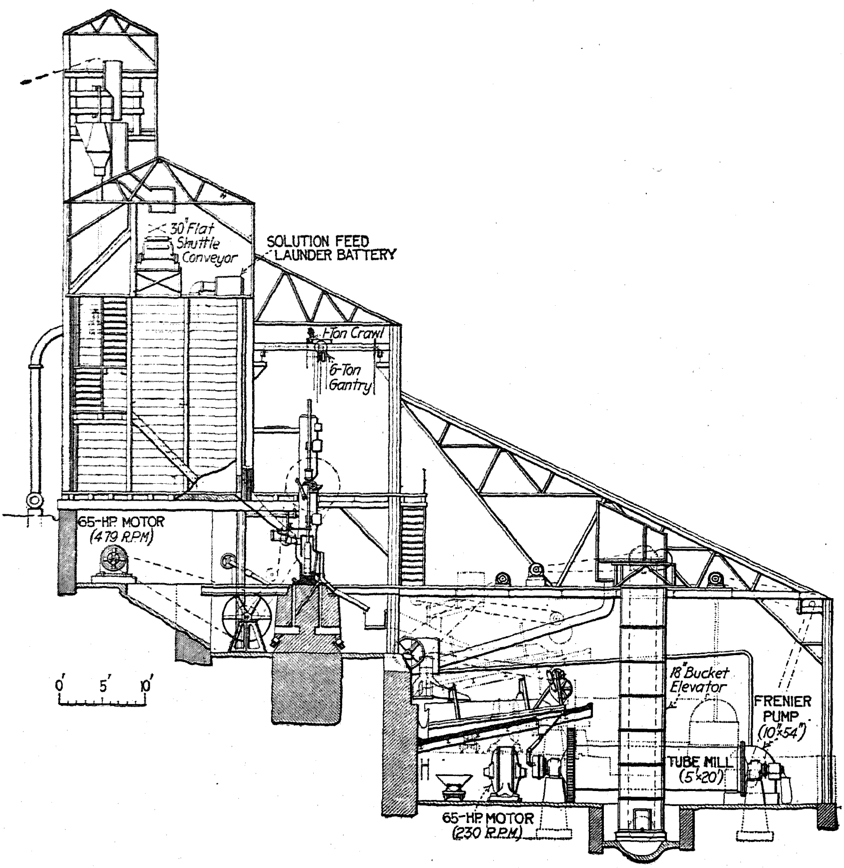


FIG. 15.—STAMP MILL. SECTIONAL VIEW.

Fig. 15, thereby permitting a broken bolt to be removed easily; none has broken thus far. A traveling crane, as well as a crawl, installed with the building, was found very useful in erecting the battery and in making subsequent current repairs.

The stamp duty averages 21.1 tons through 3-mesh and 4-mesh screens. Crushing is in mill solution, 10 parts of solution to 1 of ore. Screen wear is of no importance. Steel wear per ton is: Shoes 0.16

lb., dies 0.08 lb., liners 0.60 lb. Life of shoes, 9¼ in. diameter by 14 in. long, averages 90 days; of dies 97 days; of liners 43 days. Shoes and dies are forged steel. Liners are cast iron made locally. Manganese-steel liners have been tried, but the cost per ton crushed was considerably higher. The 65-hp. motors, each driving 20 stamps, are overloaded about 8 per cent.

A trial was made of introducing the battery solution through nozzles in the back of the mortar box above the dies, but without success. Stationary screens placed between the feeder and the mortar box to take out fines were also tried and abandoned, the benefit being doubtful while requiring more supervision.

Average screen analyses of feed and discharge of the stamp battery, equipped with 3-mesh, No. 32 wire screens, are:

	Feed, Per Cent.	Discharge, Per Cent.
Inches		
+ 2	0.5	
+ 1	18.4	
+ ¾	13.8	
+ ½	8.0	
Mesh		
+ 4	18.7	4.0
+ 8	11.5	16.3
+ 10	1.8	7.5
+ 20	8.5	14.8
+ 30	3.9	8.1
+ 40	1.4	4.4
+ 60	2.2	6.7
+ 80	2.0	6.2
+100	0.5	2.7
+120	0.8	3.2
+150	0.4	1.7
+200	0.2	2.7
−200	6.9	21.2
	99.5	99.5

Distribution of the pulp from the battery to the primary classifiers by means of split launders could be improved by a mechanical distributor of one of the successful revolving types.

The classifier platforms were built on an incline to save headroom over the tube-mill gears as well as to follow the slant of the classifiers. Returns to the classifiers from the tube mills are by elevators with individual Frenier pumps as reserves. A better design following later practice would be to effect such returns by means of the classifiers themselves.

Tube-mill grinding is done in two stages, using 5-ft.-diameter mills

throughout, the primary series being 16 ft. long and the secondary series 20 ft. and 22 ft. Comparison of this system with single-stage grinding has failed to show conclusive results in its favor, although a slight benefit is apparent. This benefit is, however, insufficient to warrant a repetition of this refinement of design unless in conjunction with water concentration not required with this ore.

Danish flint pebbles were used for a considerable period, but their increasing cost led to the adoption of mine rock entirely to replace the pebbles. The mine rock supply is obtained mechanically in the crushing plant as described and is sent separately over the regular conveyors to a compartment in the battery bin from which it is transferred by chute to the primary tube-mill floor where it is distributed by car. Part of the rock is introduced into the mill through the feeder. As the trunnion opening of the mills is not as large as it should be, rocks over 5 in. in size as well as occasional large boulders, 12 to 15 in., required in the primary mills for efficient grinding, are loaded into the mills through the manholes once a day; 130 lb. of mine rock are required for each ton of ore milled and are credited to the total tonnage treated.

Tests are now in hand using cast-iron balls in place of mine rock. Results thus far obtained indicate a capacity increase of 33 per cent. with finer grinding. Power load shows an increase of 33 per cent., from 65 to 90 hp. per mill. Forged-steel balls ordinarily used for such grinding were not obtainable but it is quite probable that chilled cast-iron or semi-steel balls and liners will prove more economical, taking into account the low cost of locally made castings, 2.5 c. per pound, as against high first cost plus importation expenses of steel balls. Ball wear is 1.7 lb. per ton milled. Tube-mill liners are modified El Oro type, of hard cast iron, the average life being 6 months; cost 2.2 c. per ton of ore milled.

The motor driving each mill through a flexible coupling and one reduction of spur gearing is of high torque induction type, 65-hp., 230 r.p.m. Allis Chalmers make. Its ample design easily carries the overload. But the power factor of this motor is low, 75 per cent., and an improvement would be to use a higher-speed motor with a single reduction of herringbone gearing.

The economical grinding point is taken at 75 per cent. through 200-mesh. Table 2 gives screen tests of feed and discharge of primary and secondary tube mills, using mine rock and at a plant capacity of 1,000 tons per day.

Agitating

An extraction of 55.5 per cent. of the gold and 18.6 per cent. of the silver takes place in the mill before the pulp reaches the Pachuca tank system.

TABLE 2.—*Screen Tests of Tube-Mill Feed and Discharge*

Mesh	Primary Feed, Per Cent.*	Tube Mill Discharge, Per Cent.	Secondary Feed, Per Cent.†	Tube Mill Discharge, Per Cent.
+ 4	13.9	0.2	0.8	1.6
+ 8	22.8	4.5	0.5	0.1
+ 10	8 6	2.2	0.4	0.2
+ 20	20.6	8.5	1.6	0.2
+ 30	11.8	10.9	3.4	0.7
+ 40	4.7	4.8	3.3	1.0
+ 60	8 0	12.1	17.5	10.8
+ 80	1.9	10.2	9.6	5.6
+100	1.6	5.2	11.4	10.3
+120	1.5	6.5	17.6	15.5
+150	0.7	4.0	9 5	11.8
+200	0 8	5.7	8.4	11.2
−200	2 5	25.0	15.6	30.7
	99.4	99.8	99.6	99.7

* Moisture, 35 to 40 per cent. Operating without return, 175 tons of ore pass through tube mill per 24 hr.

† Moisture, 35 to 40 per cent. Operating with return, in closed circuit, 200 tons of ore pass through mill per 24 hr.

Secondary Classifier Overflow

(Finished Product of Mill)

Mesh	Per Cent.
+100	3.8
+120	5.7
+150	5.5
+200	9.5
−200	75.1
	99.6

Cyanide of sodium, either 128 or 120 per cent. as obtainable, in lump or in brick form, is added at the first tank of the primary series, at the rate of 4 g. sodium cyanide for each gram of silver in the ore delivered to the mill. The sodium cyanide consumed is 3.15 lb. per ton, including mechanical loss. The strength of solution at the beginning of agitation is 0.55 per cent. KCN; of the mill solution, 0.4 per cent.

Protective alkalinity is maintained at about 0.75 per cent.; the lime, fed dry into the ore at the crushing plant, is low-grade, averaging about 65 per cent. available, the consumption being 20 lb. per ton. Arrangements are in hand to improve the method of feeding by emulsifying the lime, adding it either to the primary tube mills or to the Pachuca tanks.

Crude litharge, between 85 and 90 per cent. PbO, adopted in place of lead acetate as both cheaper and more efficient, is ground in a small

tube mill, 24 in. diameter by 37 in. long, discharging into the first Pachuca tank at the rate of 0.6 lb. per ton of ore.

The best results are obtained with 72 hr. agitation although 60 hr. give within 2 per cent. as high extraction. The air pressure is 27 lb., 75 cu. ft. per minute being required for each tank.

No difficulty has been experienced with the series operation of the Pachuca tanks. Connections between tanks are made by 10-in. horizontal pipes, located 3 ft. from the top, the joint between two abutting pipes being made leak proof by a wrapping of tarred canvas. This joint takes up the tank vibration and effects an easy connection.

To avoid undue accumulation of slime on the inside of the tanks, they are emptied and sluiced down once a month. This operation requires about 3 hr. per tank, using a 3-in. Traylor slime pump for the return.

Screen tests of the inflow and outflow of the system are practically identical, showing that there is no segregation or short-circuiting of sand or slime.

Filtering

This step in the process is fairly difficult, due to fine grinding of ore containing a considerable amount of colloidal matter. Also, the dissolved values in the pulp to the filters are high, 3.5 oz. silver per ton of solution. After extended working-scale tests of several types of vacuum and pressure filters, Merrill presses were adopted, using the center system of filling.

A press cycle occupies 75 min. made up of:

	Min.
Charging	26
Barren solution wash	13
Water wash	13
Sluicing	23
	75

Bristol recording pressure gages are attached to the filling pipe of each press, the cycle curves from the chart giving an excellent check upon the care taken by the attendants in operating the presses. The sluice valve bar and the filling valves are electrically connected so that both cannot be coincidently opened without ringing an alarm bell. The cakes average 1¼ in. in thickness, thereby leaving ½ in. space in the center of the 3-in. frame for entrance of washes. Dry pulp handled per cycle, 14.5 tons.

Sluicing water is at 90 lb. pressure and five parts are required to clean out a press. The bulk of this water is recovered in dewatering tanks as explained heretofore.

No. 6 cotton duck, 72 in. wide, is used regularly, after extensive trial of several other weights. A set of cloths lasts about 2,500 cycles. Acid

washing to remove lime is done in the press every 10 days, using a 0.75 per cent. sulphuric acid solution.

Much care is taken with the nozzles of the sluice bar. The bar of each press is taken out and tested every 48 hr., any defective nozzles being replaced. Nozzles are ordinary ½-in. cast-iron plugs, drilled 3/16-in. hole; these last practically as long as special steel nozzles and are very much cheaper.

The unwashed values in the press discharge average a trace of gold and 0.08 oz. silver, showing an efficiency of 97.7 per cent.

Clarifying Solutions

All solutions to be precipitated are first passed through ordinary sand filter tanks. The addition of about 40 per cent. by volume of sawdust to the sand considerably improves the clarifying efficiency of the filtering medium and at the same time reduces the frequency of slime skimming and sand cleaning. The latter operation is effected by shoveling the sand from the tank to a launder leading to a small trommel where the sand is washed free of slime, with small loss, and returned for re-use.

In this way, 4,500 tons of solution are filtered daily at a cost of 0.28 c. per ton of solution. Possibly Merrill clarifying presses would be an improvement, although no comparative figures are available. At any rate, the existing equipment does the work efficiently and cheaply, with little supervision required.

Precipitating

The Merrill zinc-dust process of precipitation has in every way justified its adoption. Its efficiency, safety, cleanliness, etc., compared to the zinc-shaving method are too widely known to need further elucidation here. The ease and rapidity with which a cleanup can be made gives it a further special advantage in a good-sized mill treating silver ores with the consequent considerable production of precipitate. Formerly the process enjoyed the further advantage of a lower price for zinc dust than for zinc shavings; the difference now is practically negligible.

The solution from the sand filters averages 0.015 oz. gold and 3.0 oz. silver. Two circuits are maintained, one precipitating to barren solution, sufficient for the filter press washing, and the other circuit to an effluent containing from 0.9 to 1.2 oz. silver per ton. To accomplish this a slight excess of zinc dust is fed to the barren circuit in the proportion of 1.1 oz. zinc dust to 1.0 oz. fine bullion; to the partial circuit 0.8 to 1.0. The average zinc consumption of the plant is 1.0 oz. to 1.0 of fine bullion.

Cleanups made at the middle and end of month and sometimes oftener, provide an accurate check on current work.

Melting

The only drying given the precipitates is by blowing air through the presses at 25 lb. pressure for 1 hr., reducing the contained moisture to about 30 per cent. The precipitate is cleaned from the presses into rectangular, steel cars, 5 ft. 10 in. by 6 ft. 4 in. by 1 ft. deep, in which it is weighed. Based upon the calculated dry weight of the precipitate, flux consisting of 1½ per cent. each of borax and bottle glass, is added on top without mixing, and the charge is then shoveled from the car into No. 400 crucibles. To permit the introduction of a high column of precipitate, a discarded pot with the bottom out is temporarily fitted into the crucible, being removed after the charge has melted down. This procedure reduces dust losses and accelerates charging and melting.

The eight furnaces (Fig. 16) are oil-fired, built flush with the floor of the melting room, on the arc of a circle with fire pit at rear. This arrangement has proved very satisfactory and is to be recommended as convenient and labor saving.

The crucible, upon being lifted from the furnace by tongs attached to ½-in. wire cable raised or lowered by air cylinder on the post of the jib crane, is swung around to one side to the pouring carriage containing the bullion molds. The slag is first skimmed off, using sand to assist in this operation. As the bars are poured, sticks of wood are laid on top and, igniting, prevent too rapid cooling of the center of the bar with the consequent subsidence and holing.

The precipitate averages 85 per cent. fine bullion; the Doré bars weigh 1,000 oz. and assay 5.0 gold and 940 silver fine. The No. 400 crucibles contain an average of seven bars per melt with a maximum of 10 bars. The average life of a crucible is 10 melts or 70 bars.

With bullion of this fineness, no difficulty with matte formation is experienced. The bars are carefully cleaned and all chips and corners removed before sampling and weighing. Drill samples are taken, one on the top of the bar, one-third the distance along the diagonal toward the center, and one on the bottom of the bar at one corner; each hole is drilled halfway through the bar.

The bullion was formerly refined by a French concern in Mexico City, the fine gold being sold to the Mexican Government and the fine silver shipped for sale in New York or London. The refinery has been closed down since the outbreak of the war in Europe, so that the Doré bars are now refined and sold in New York or London.

Melting the precipitate in an electric furnace, built after the Alaska-Treadwell and Lluvia de Oro design, is under trial. Alternating current at 120 volts is used, the input being about 600 amp. working at full heat. Trouble has been experienced with dusting and with burning a hole through the bottom of the furnace; a renewable iron plug (Fig. 17) may

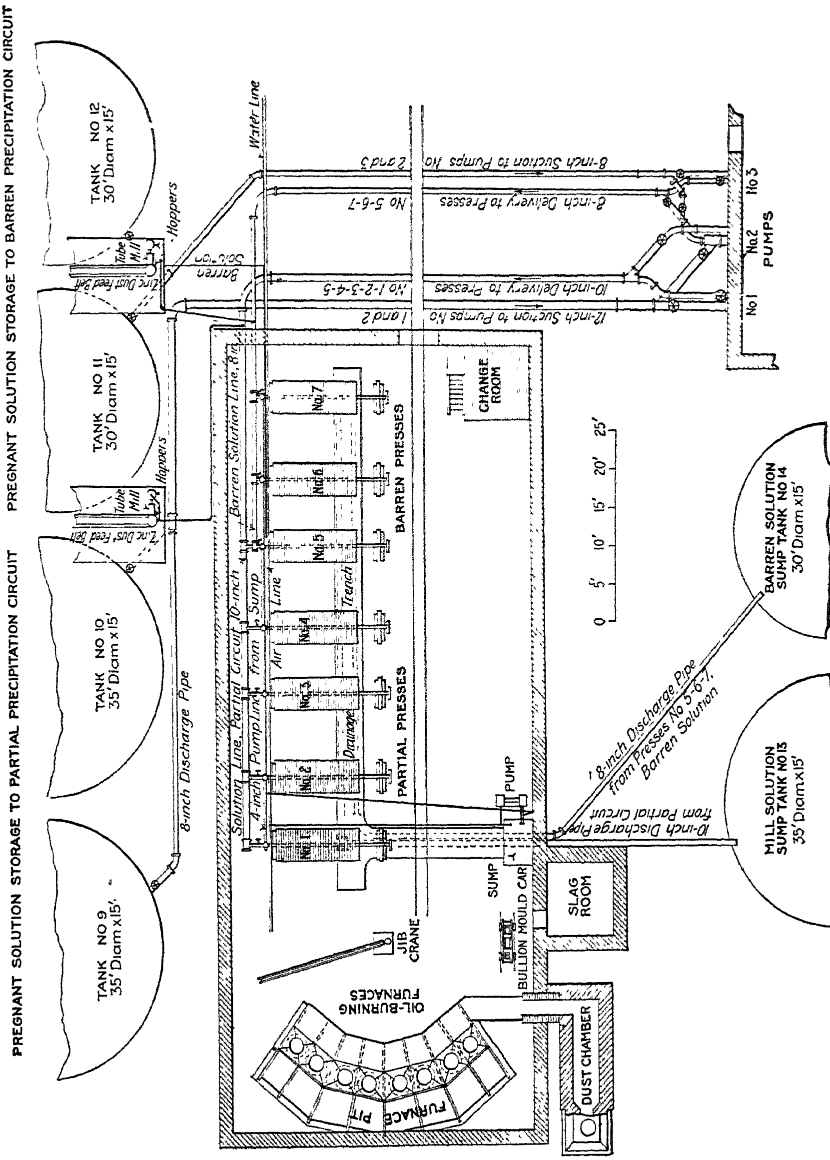


FIG. 16.—PLAN OF PRECIPITATING AND MELTING ROOM.

provide a convenient remedy for the latter. The dusting is probably due to the precipitate being wet, the intense heat at the electrode setting up a center draft of steam and air. Either the precipitate should be dried or some dust-collecting apparatus attached to the furnace top. The advantages of this system over the oil-fired crucible method of melting are rapidity of melting and discontinuance of the use of crucibles and oil, both of which require a long distance haul; no data are available yet as to the comparative economy.

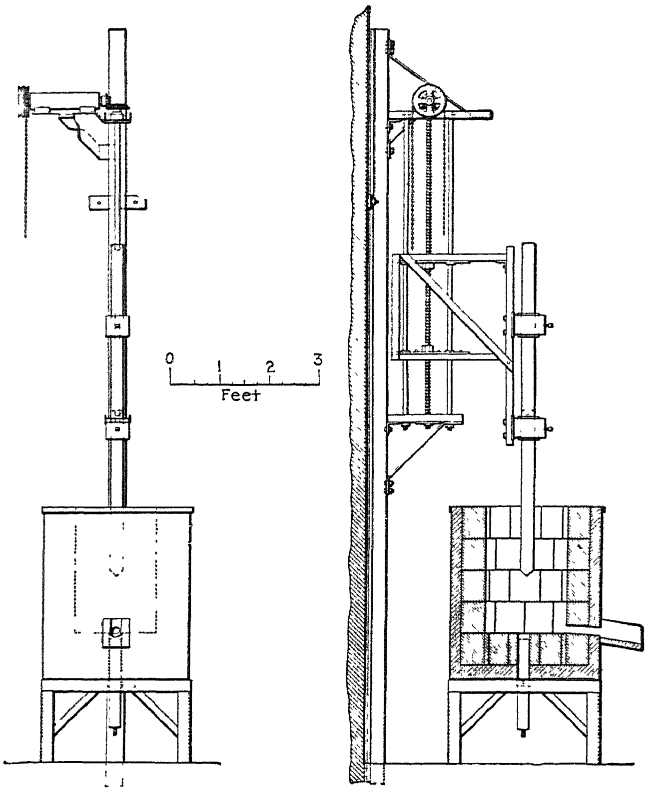


FIG. 17.—ELECTRIC MELTING FURNACE.

Tailing Disposal

As previously described, the tailing sluiced from the Merrill filter presses passes through four Dorr thickening tanks for recovery of water and is then automatically sampled before flowing to the storage basins. These basins or ponds, four in number, are located on the nearest flat land miles away. The ditch thereto was dug U-shaped, 2½ ft. deep with a per cent. grade; 1½ per cent. grade would suffice.

The walls of the storage basins are built up as required of tailing mud,

one basin receiving the entire inflow while the others are being prepared. Evaporation and particularly absorption of the solution are high, the decanted effluent representing only 30 per cent. of the solution inflow.

This solution contains an average of a trace of gold and 0.12 oz. silver, representing filter losses and further dissolved values. A small precipitation plant is being arranged to recover this escaping fraction.

Construction Costs

On the basis of 1,100 tons daily capacity the total cost of the plant was $792 per ton capacity. This covered first-class construction methods throughout, designed for long life with minimum maintenance and renewals. A comparatively excessive amount of equipment was required due to fine grinding and especially to length of treatment necessary for a silver ore, resulting in unusually large tankage. On the other hand, the all-sliming design without concentration simplified operations and eliminated the installation of concentration equipment.

Experimental Work

Aluminum Dust Precipitation[3]

Upon the publication of results obtained at Nipissing with desulphurization and aluminum-dust precipitation, similar experiments were undertaken. Desulphurization, as was to be expected, gave no beneficial results on Santa Gertrudis ore, which contains no tellurides, arsenides, etc. On the other hand, aluminum-dust precipitation showed a considerable improvement over the zinc-dust method and its adoption has only been delayed by war conditions practically having put aluminum dust off the market. Comparison of the two methods was based upon normal prices of aluminum and zinc dust of 30 c. and 7 c. per pound, respectively.

As practically all the existing equipment can be used very conveniently in the aluminum-dust process, the problem as worked out depends for commercial success upon an economical method of converting the lime in solution, added during milling for neutralizing and settling, into calcium carbonate with the formation of the necessary caustic soda as a product of this reaction, all prior to precipitation by aluminum-dust. Any lime present in solution during precipitation will be converted to calcium aluminate, lowering the grade of the precipitate and causing difficulty in melting. The conversion and elimination of lime can be

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[3] Tests conducted at mill by A. W. Hahn.

accomplished by the addition of carbonate or bicarbonate of soda, as follows:[4]

$$Ca(OH)_2 + Na_2CO_3 = CaCO_3 + 2NaOH$$
$$Ca(OH)_2 + NaHCO_3 = CaCO_3 + NaOH + H_2O$$

In place of commercial carbonate or bicarbonate of soda, laboratory tests showed the probable successful use of an impure local product known as "Tequesquite," obtained from the dried-up bed of Lake Texcoco and containing approximately 56 per cent. carbonate and bicarbonate of soda in about equal quantities. In practice, the "Tequesquite" solution would be added to the pulp flowing to the collecting tanks above the Merrill filter presses.

In experimental work, the aluminum-dust precipitate yields a bullion 975 fine. The consumption of aluminum dust was about one-third that of zinc dust, or 0.34 oz. to 1.0 oz. fine bullion.

The slightly increased cost per ton of ore of aluminum dust is more than offset by decreased chemical consumption of cyanide, the net saving being calculated at $0.15 per ton of ore.

Electrolytic Regeneration of Cyanide[5]

Prior to the favorable conclusions reached in the aluminum-dust tests, a lengthy investigation was conducted in the regeneration of cyanide in the mill solution by electrolysis.

Anything like a comprehensive description of this interesting work cannot be taken up here and only the important features will be outlined.

The principal difficulty presented was in finding an anode of high conductivity that would not dissolve or disintegrate. Graphite, lead, plumbago, carbon, charcoal, magnetite, peroxidized lead, were tried without much success. An alloy of lead with 6 to 9 per cent. antimony was finally selected as fairly satisfactorily filling the requirements. The following conclusions were then worked out:

1. The solution to be treated by electrolysis must be barren of gold and silver to avoid their precipitation. The question of enlarging the

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[4] For chemistry and details of aluminum-dust process refer to:

E. M. Hamilton: Aluminum Precipitation at Nipissing, *Engineering and Mining Journal*, vol. 95, No. 19, p. 935 (May 10, 1913).

E. M. Hamilton: *Engineering and Mining Journal*, vol. 99, No. 13, p. 568 (March 27, 1915).

S. F. Kirkpatrick: Aluminum Precipitation at Deloro, Ontario, *Engineering and Mining Journal*, vol. 95, No. 26, p. 1277 (June 28, 1913).

James Johnston: The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, *Trans*, vol. 48, p. 3 (1914).

G. H. Clevenger: The Mill and Metallurgical Practice of the Nipissing Mining Co., Ltd., Cobalt, Ont., Canada, *Trans.*, vol. 49, p. 156 (1914).

[5] Tests conducted at mill by A. W. Hahn.

scope of the process to include electrolytic precipitation of gold and silver was also investigated separately, as hereinafter discussed.

2. The regeneration of cyanide takes place at the cathode, with a destruction of cyanide at the anode which may be principally overcome by maintaining a high protective alkalinity at the anode.

3. A porous diaphragm such as canvas separating the anode and cathode assists in maintaining an alkaline anolyte, circulation of anolyte and catholyte being accomplished individually.

4. Power consumption is affected by resistance of electrodes, electrolyte and diaphragm, by temperature and circulation rate of electrolyte, and by spacing of electrodes.

5. The voltage drop per cell, at a current density of 15 amp. per square foot of anode surface, treating barren solution containing 0.5 per cent. total KCN and 0.115 per cent. CaO, with canvas diaphragm and electrodes spaced 1½ in. apart, is 6 volts at 70° F. Under these conditions the solution flow should be at least 0.25 gal. per square foot of anode surface.

6. The deposit on the cathode analyzed 71 per cent. zinc, with impurities of copper, iron, arsenic and organic matter. The recovery of this zinc might be turned to some commercial advantage.

7. The reaction representing the regeneration of cyanide is presumably as follows:

$$Na_2Zn(CN)_4 + Ca(OH)_2 + \text{Current} = 2NaCN + Ca(CN)_2 + Zn + H_2O + O.$$

8. The net reduction of cost was estimated at $0.12 per ton of ore. The equipment outlay, however, proved to be several times greater than that required to convert the precipitation plant to the use of aluminum dust and as the latter process at the same time indicated a somewhat larger cost reduction without further complexities of plant, its adoption in place of regeneration was decided upon.

Electrolytic Precipitation and Regeneration

Following the experience gained in the experiments with regeneration above described, the next step was to endeavor to combine with it the electrolytic precipitation of gold and silver from solution.

Difficulty was encountered in obtaining a coherent cathode deposit. It was found uneconomical to attempt a complete precipitation, the limit of commercial work being about 1.3 oz. silver remaining in solution. Plant cost would be high. Altogether the proposition did not present sufficiently attractive possibilities to warrant following it up very thoroughly. With the perfection of the diaphragm cell it is probable that both precipitation and regeneration electrolytically could be worked out to a fair success.

Electrolytic Refining of Precipitate

A preliminary investigation has been made of electrolytic refining of precipitate from the present zinc-dust installation. The chief difficulties to be overcome are resistance of the precipitate to the current, purification of the electrolyte, high acid consumption due to impurities, and refining and melting of the gold slime.

Flotation Concentration

As previously stated, the ore is clean with only small quantities of base metals. A representative analysis is: Au, 0.067 oz.; Ag, 12.0 oz.; Pb, 0.10 per cent.; Cu, 0.20; Zn, 0.15; Fe, 4.40; SiO_2, 86.67; S, 1.0 per cent.

Most of the silver is present as argentite, Ag_2S – 0.05 per cent. It is evident, therefore, that little benefit in recovery is to be expected by gravity concentration. At the time of building the plant complete experiments were conducted on a working scale, the recovery by concentration representing from 8 to 10 per cent. of the value of the ore. This did not justify the installation of gravity concentrating equipment plus added cost of operation as against direct cyaniding.

The growing success of flotation concentration in the United States led naturally to local investigation of the problem, which though still in a laboratory stage, already shows encouraging results. If the process is worked out to a commercial success, a detailed description of results should be of interest. Meanwhile a summary of preliminary conclusions is given:

1. A large variety of vegetal and mineral oils have been tested, the best results thus far having been obtained with one-third pine oil or wood creosote and two-thirds Mexican gas oil, such as is used locally in the oil-fired melting furnaces. The pine oils and wood creosote tried are imported, but doubtless satisfactory substitutes can be made near at hand. Several coal tars and a number of Mexican oil field products remain to be tested.

2. The addition of acid shows no particular benefit. A slight alkalinity not exceeding 0.03 lb. lime per ton solution is satisfactory.

3. Heating the solution does not materially improve results.

4. The flotation tailing is cyanided readily, the presence of the small quantity of oil having no deleterious effect.

5. The raw flotation concentrate cyanides with no more difficulty than a gravity concentrate; the cyanide consumption maintains more or less the same ratio as in the crude ore, viz.: 4 g. sodium cyanide per gram of silver; a 98 per cent. extraction of gold and silver in the concentrate is obtainable.

6. In the small laboratory machines without cleaning cells, a recovery by flotation of 65 to 70 per cent. can be made at a ratio of concentration of between 80 and 90 to 1.

7. In the 100-ton trial plant now under construction, equipped with middling returns and cleaning cell, it is expected that these results can be improved.

8. It is possible, though not certain, that flotation, even with this clean ore, may replace cyanidation to a large extent.

Power

The Pachuca district is supplied with electric power generated by several hydro-electric plants owned by three power-distributing companies, practically under one control. The main plant, owned by the Mexican Light & Power Co. and located at Necaxa, Puebla, about 75 miles by transmission line to the east, also furnishes energy for Mexico City and El Oro. The 50-cycle alternating current is transmitted at 85,000 volts to the Pachuca substation where it is stepped down to 20,000 and 6,000 volts for local distribution. The power is sold at the substation of the purchaser on the basis of integrating wattmeter readings, no peak or installation loads being considered. A power factor of at least 80 to 85 per cent. is required of the purchaser and prices vary from $42.50 to $50 per horsepower-year, depending on amount consumed.

Although most of the mines are wet, there are no reserve steam plants in the district, with the exception of a relatively few old steam hoists that could be used for baling in an emergency. Nevertheless, the several widely separated hydro-electric plants with individual transmission lines provide a very reliable power supply.

General Remarks

As can be imagined, operating conditions in Mexico during the past several years have been subjected to many unusual difficulties, although the Pachuca district perhaps has been as fortunate as any in maintaining a fair rate of work. This has been due principally to its advantageous geographical location, numerous railroad connections and steady power service. Added to these favorable factors, there has been coöperation among the mining companies and a considerable resourcefulness in looking after the labor and in bringing in supplies.

Like other countries at present under strain, Mexico has found it necessary to increase all taxes very materially, including those on mining property and on metals produced and exported. Very luckily, the prevailing high metal prices still enable the silver and base-metal producers to make a profit, but the gold-mine owner is hard hit with no rise in the market price of his product to offset steadily increasing costs as affected by the taxes, higher prices for supplies, etc.

The labor outlook is not encouraging. As a natural outcome of several years of revolutions, labor is also passing through a similar period of unrest, and much tact combined with fair and firm treatment will be

necessary to meet this situation in any way successfully. The conditions are likely to be accentuated, due to a shortage of labor supply when mining activity throughout Mexico is again possible.

Discussion

Jay A. Carpenter, Tonopah, Nev. (communication to the Secretary*).—This description of the Santa Gertrudis mill is of great interest to the operators of similar silver mills in Nevada. At the San Francisco meeting last September there were two excellent papers presented on Nevada silver mills; one by A. H. Jones on The Tonopah Plant of the Belmont Milling Co., and the other by E. E. Carpenter on Cyaniding Practice of the Churchill Milling Co., Wonder, Nev. At the same meeting, I presented a paper on Slime Agitation and Solution Replacement Methods at the West End Mill, Tonopah, Nev., covering this particular feature of our milling practice. A comparison of the data presented in these four papers is interesting in that, while the general design and practice is much the same in all cases, there is a wide difference in the design and practice in the various departments of the mills.

It is to be regretted that Mr. Rose did not round out this excellent article by following Mr. Jones's and Mr. Carpenter's lead in giving the average gold and silver content of the ore milled and the resulting tailing, with the consumption of chemicals and the cost of milling.

The practice of the Santa Gertrudis mill in the use of cyanide and zinc offers an interesting comparison with that of the Nevada mills, and a discussion of this point may bring out the reasons for the differences that exist.

Mr. Rose describes the ore as "clean with only small quantities of base metals." The fact that the zinc dust precipitate is 85 per cent. gold and silver, and that melting with only 3 per cent. of fluxes the bullion is 945 fine, points to the fact that the bases present in the ore are quite inert to the action of the cyanide solutions. In fact, the ore appears to be very similar to the Tonopah and Wonder ores; yet the strength of the cyanide solution in the mill and the amount of cyanide used per ounce of bullion is nearly double that of the Nevada plants, while the zinc consumption is about the same.

The figures given in the accompanying tables are from articles in the *Transactions*, except those of the West End mill, which are taken from the annual report and mill data for 1915.

In the accompanying tables the gold and silver content of the Santa Gertrudis ore is given the same as Mr. Rose gives under "a representative analysis" in discussing flotation, and the ounces gold and silver precipitated are assumed, since extraction figures are not given, at a little above 90 per cent. of the ore content, which corresponds closely to Tonopah

* Received Sept. 21, 1916.

results. The total time of the ore in the different mills is roughly figured from the total ore tankage in the mills, and the figures are relatively correct.

These two tables give considerable data that should be of special interest to the students of cyanide solutions and the cyaniding of silver ores.

S. J. Kidder, Mogollon, N. M. (communication to the Secretary*.)—Mr. Rose says that in clarifying their solutions they pass all solutions to be precipitated through sand filter tanks and that the addition of about 40 per cent. by volume of sawdust to the sand considerably improves the clarifying efficiency of the filtering medium. In this connection the writer would like to ask Mr. Rose if such use of sawdust is not liable to cause serious loss unless the sawdust is finally recovered and burned and the ashes saved? In the larger silver plants in Nevada it was the general experience that wood shavings, sawdust, cotton waste, sticks and rubbish in general which contained carbonaceous matter, if in contact with pregnant solutions for any length of time, would invariably cause considerable amounts of silver and gold to be precipitated. For this reason it was the usual practice to burn all such material and save the ashes, which were shipped from time to time to the smelters with either concentrates or slag. It would seem that the addition of 40 per cent. by volume of sawdust to the sand filter through which pregnant silver-gold solutions were to be passed would certainly result in a decided loss by precipitation unless the sawdust was saved, which Mr. Rose does not mention.

Hugh Rose (communication to the Secretary†).—Carefully taken samples of this product after a year of service assayed about 120 grams silver per ton, this being more or less the grade of solution passing through clarifiers. These same samples washed with distilled water in a small vacuum filter, where there is no possibility of washing out anything but dissolved values, reduced to 6 grams silver per ton. Were any precipitate present, it would not be soluble in water and would appear in the re-assay after washing. While there is a remote possibility that precipitation might take place, the chances are very small compared to the risk we run with the large amount of decayed timber brought into the mill every day in the ore flow, and from which up to the present time we have been unable to notice any bad effect. We have frequently had decayed wood pulp collected from various parts of the plant, principally from the foam on the top of the secondary Dorrs, where there would be every opportunity of considerable precipitation and enrichment, but with so far no sign of precipitation. The contained values run approximately what the solution assays, and a water wash will in each case remove them, proving that they are not there as metallic or precipitated values.

* Received Sept. 5, 1916.
† Received Oct. 26, 1916.

Tables Accompanying Discussion by Jay Carpenter.

Cyanide Data

Mill	Contents of Ore		Per Cent. Values Removed by Concentration	Strength of Solution		Consumption Sodium Cyanide		Hours of Ore		Temperature of Solutions
	Oz. Au	Oz. Ag		At Batteries	At Start of Agitator	Per Ton Ore	Per Fine Oz. Bullion Recovered	In Agitators	In Mill	
				Lb. KCN	Lb. KCN		Lb.			
Wonder...........	0.216	19.48	None		4.5	1.72 lb. NaCN	0.095	60	120	About 90°F.
Belmont..........	0.234	22.34	About 12	5.0		3.5 lb. KCN or 2.71 lb. NaCN	0.153	48	96	90°F.
Santa Gertrudis....	0.067	12.00	None	8.0	11.0	3.15 lb. NaCN	0.280	50	96	Presumably 90°F.
West End..........	0.245	23.42	None	3.4	3.7	2.60 lb NaCN	0.120	70	110	115°F.

Zinc Data

Contents of Ore the Same as in Table Above

Mill	Method of Precipitation	Oz. Au and Ag Precipitated per Ton Ore	Approximate Tons Solution Precipitated per Ton Ore	Oz. Au and Ag Per Ton Solution	Oz Ag in Tails		Lb per Ton Ore	Zinc Used per Oz Au and Ag in Bullion	Per Cent. Au and Ag Precipitate	Per Cent. Fluxes Added	Fineness of Bullion
					Partial	Complete					
Wonder...........	Thread	18 15	6.8	2.7	0.12	0.12	1.63	0.089	63.5	About 15	968
Belmont...........	Dust	17.67	6.0	2.6	0.16	About trace	0.95	0.054	80.6	14	932
Santa Gertrudis....	Dust	11.00	4.5	3.0	1.00	About trace	0.69	0.063	85.0	3	945
West End..........	Thread	21.70	3.6	6.0	0.40	0 07	1.21	0.056	65 0	15	930

Cyaniding Clayey Ore at Buckhorn, Nevada

BY PAUL R. COOK,* B. S., GUAYAQUIL, ECUADOR

(Arizona Meeting, September, 1916)

THE ore deposit of the Buckhorn Mines Co., Buckhorn, Nev., is peculiar in being a shallow kaolinized mass of material with basalt walls, and having apparently no direct connection with any of the usual gold-bearing rocks. The average ore contains 16 per cent. water of hydration, and the cyaniding of this hydrous clayey material offered unusual difficulties as compared with an average Nevada quartz ore.

The orebody was thoroughly developed; then the mill was built according to the latest cyanide practice, with such changes as were thought to be demanded by the peculiar nature of this ore.

Upon starting the mill, the ore proved more difficult to handle than had been anticipated. It is hoped that an account of how these difficulties were met may prove of interest to anyone having a clayey ore to handle and to the profession in general.

Geology

The Buckhorn orebody lies along a north and south fault plane of perhaps 1,000 ft. dislocation, that can be traced for miles; but the only other known mineralization consists of similar ore in the Murphy mine, a mile farther north.

The east or hanging wall is hard and smooth, being a typical fault plane. The best ore is along this wall, gradually grading down toward the west until at 30 to 60 ft. it is too low-grade to mine. The country rock on the west consists of alternating layers of hard and soft basalt and basalt scoria, pitching toward the mine.

One of these basalt layers on the hillside a little above the mine is marked, for 3 or 4 miles in length, by a line of springs which seep out along it. Perhaps the meeting of the surface drainage, passing down these basalt layers, with the fault-plane solutions explains the formation of the Buckhorn orebody.

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* With Buckhorn mines during the first year of mill operation (December, 1913, to December, 1914).

Beneath the oxide ore a smaller body of almost pure marcasite occurs with about the same assay value ($8 per ton). Beneath the 250-ft. level there appears to be no further mineralization.

Mining

The first difficulty was to get the ore out of the mine. Since the orebody extended from the grass roots to a depth of 175 ft., with a width of 50 to 80 ft. and a length of 1,400 ft., the "glory-hole" system of mining was adopted. The ore was broken through 8-in. grizzlies into 18 chutes extending from the surface to an intermediate level, on which it was trammed by hand to three chutes leading to the 200-ft. level, and hauled by electric motors through a 1,000-ft. tunnel to the mill.

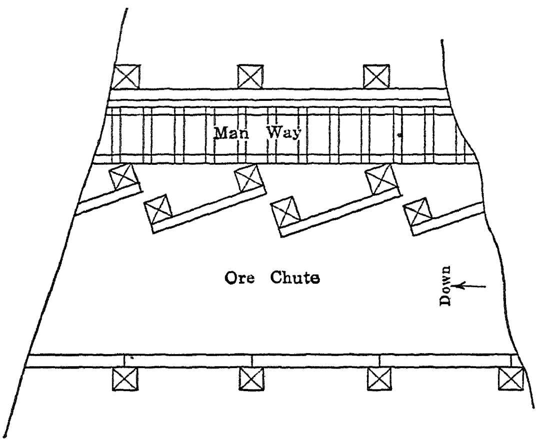


Fig. 1.—Section through Ore Chute.

In the summer little trouble was experienced; but the ordinary rain and snowfall of a Nevada winter made it almost impossible to keep the chutes open, as first built. After reconstructing, so as to provide an opening for barring at each set of timbers (see Fig. 1), it became possible to keep the mill supplied with ore during all kinds of weather. During wet weather all the cars had to be scraped with a shovel after dumping, and unusual care was required to keep a sticky load from carrying car and all into ore bin or over the dump; 30 per cent. of the tonnage mined went to the waste dump.

Milling

Ore Bin and Crusher

The next problem was to get the ore out of the mill bin and crushed. The bin was an ordinary circular steel bin, with natural earth bottom and

side gate. This ore absolutely refused to run from the bin. The mill was built to treat 300 tons a day, but even with one man in the ore bin, and two at the crusher, it was impossible to get over 150 tons through in 24 hr. The large kaolin lumps gave the most trouble in crushing. They had to be practically chiseled to pieces and poked through the jaw crusher by hand.

The replacement of the jaw crusher with a high-speed toothed roll (see Fig. 2) gave the desired crushing capacity. This machine was developed at one of the Bingham Canyon (Utah) properties, and is manufactured by a Salt Lake firm. It is well adapted for sticky ores. The Buckhorn ore contains an occasional "nigger head" of very hard "malapi" or basalt. The mill crew was afraid one of these would break off the teeth of the crusher shell, and they were very carefully picked out at first,

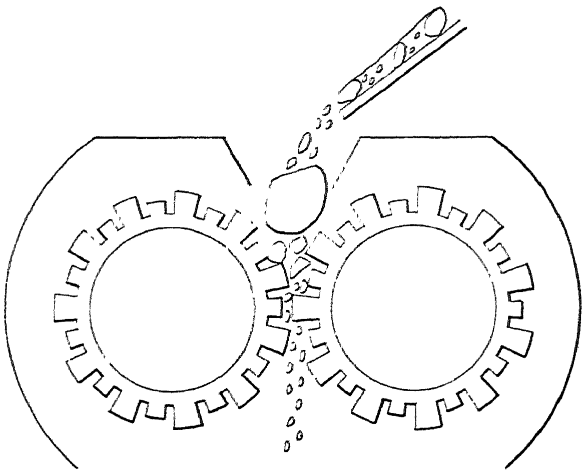


FIG. 2.—SECTION THROUGH WALL CRUSHER SHELLS.

but after a few had gone through by accident, and a few pieces of drill steel, etc., had been chopped up, no further concern was felt. All the boulders that had been sorted out were later put through as it was found that the easiest way to keep the teeth free of adhering clay, was occasionally to throw in a boulder of hard rock.

To do away with the necessity of a man on each shift to shovel the ore out of the bin, a 36-in. conveyor belt was installed to feed the crusher automatically. The opening in the bottom of the bin, over the belt, was about 2 ft. wide, and extended clear across the bin. It was closed by means of short pieces of mine rail that could be removed a few at a time, permitting the ore to be drawn from any point desired. The belt was driven from the crusher line shaft by means of ratchet and dogs. It could be started or stopped, from either floor of the crusher building, by means of a rope connected with the dogs. With these

improvements, crushing required only a part of one man's time. Two men each shift attended to crusher, rolls, two ball mills, two classifiers and two tube mills.

Rolls

The 45 by 15-in. Anaconda-type rolls with smooth shells would clear themselves fairly well, if one of the shells had a channel about 1 in. wide by ½ in. deep machined in it, but it was troublesome to keep a groove in the shells as they wore down. A corrugated or toothed shell would have been better for this ore.

Ball Mills

One 6-ft. Hardinge ball mill was intended to handle the whole tonnage. After plastering the balls to the side of the mill with clay, a few times, the shiftmen learned to tell by the sound of the mill, when it was beginning to coat up. By shutting off the feed at this time, it took only a few minutes for the coating to be ground out.

The installation of a duplicate ball mill made it possible to keep the rest of the plant going while grinding out the ball mills, one at a time, and allowed the rolls to be set coarser on troublesome ore. With a good run of ore, 300 tons per day was sometimes put through one mill.

Classification and Tube Milling

This ball mill discharge was classified in two 36-in. Akins classifiers, the sand from which was fed to two 5 by 18-in. tube mills with Komata liners. The tube-mill discharge was classified in a home-made drag classifier. The small percentage of material requiring regrinding consisted largely of fragments of the basalt "nigger heads." This material was almost as hard as the pebbles themselves and of low assay value. Occasionally it accumulated in the circuit enough to be troublesome and was thrown away. A small amount of it was thought to help the grinding. About 80 per cent. of the product delivered to the treatment plant would pass a 150-mesh screen.

Agitation

About 80 per cent. of the mill-head value was dissolved in the crusher plant. Only a trifling additional extraction could be obtained either in the mill or experimentally. The real trouble was to remove the dissolved value from the clayey pulp. Accordingly the three 32 by 14-ft. Dorr agitators were changed to thickeners.

Thickening

These three "converted" agitators settled 300 tons per day of 1 to 10 pulp, as delivered from crusher plant, to a specific gravity of 1.15. The 8 sq. ft. of settling area provided per ton of this ore settled in 24 hr. would be sufficient to settle an average Nevada quartz ore to a specific gravity of 1.33. The overflow was precipitated, and the underflow mixed with the barren solution and fed to six 36 by 12-ft. Dorr thickeners, delivering a 1.23 specific gravity underflow to the filters. The 20 sq. ft. of settling area per ton settled in 24 hr. is three times the area required to settle an average Nevada quartz ore to a specific gravity of 1.33. Primary thickeners were held with 2 ft. of clear solution; the secondary thickeners with 6 in. It was impossible to settle the raw Buckhorn ore beyond a specific gravity of 1.26, either in the mill or experimentally.

Filtering

The maximum capacity of each of the four 14-ft. diameter by 12-ft. face Oliver filters was 50 tons per day, about one-half their capacity on a Nevada quartz ore. An additional filter, 14-ft. diameter by 24-ft. face, had to be installed to handle 300 tons per day.

Dehydration

A sample of Buckhorn ore carefully dried at a temperature below 110° C. had a specific gravity of 1.9. A higher temperature gave an additional loss of 16 per cent. in weight, and entirely changed the physical properties of the ore. The dehydrated sample had a specific gravity of 2.4, and settled and filtered almost as well as a quartz ore. Dehydrating also removed the sticky milling qualities. Both samples, however, gave the same extraction with cyanide. The temperature of a laboratory electric hot plate was sufficient to dehydrate a sample nicely. As CO_2, etc., would not be driven off at this temperature, this loss in weight must be due to water of hydration.

With a cheap fuel supply, dehydration before milling would be the best treatment for this class of material. The ore would mill and classify more easily; the thickeners and filters would have normal capacity; and dissolved values would be more completely removed. The temperature of a commercial drier would dehydrate the ore with about the same fuel consumption (100 lb. of coal per ton of ore) as in removing the 18 per cent. of H_2O if it existed in the form of moisture.

The high price of fuel delivered at Buckhorn prevented the adoption of dehydration at this mill. The ore was milled raw at the cost of $1.59 per ton (see Table 1). Power cost $8 per horsepower per month.

The careful drying at a temperature below 110° C. of a number of samples, is a tedious operation even with special equipment; and unless dehydrated the samples tended to stick in the laboratory grinding machinery, so the regular moisture and assay samples at Buckhorn were dehydrated. All assay, moisture, tonnage, etc., figures are on this basis.

To compare with other ores the figures obtained by drying below 110° C. should be used. Both sets of figures are given in Table 2. Table 3 shows the mill flow sheet.

TABLE 1.—*Buckhorn Mining and Milling Costs*

Ore Milled, 10,000 Wet Tons, 8,100 Dry Tons, H_2O 19 Per Cent

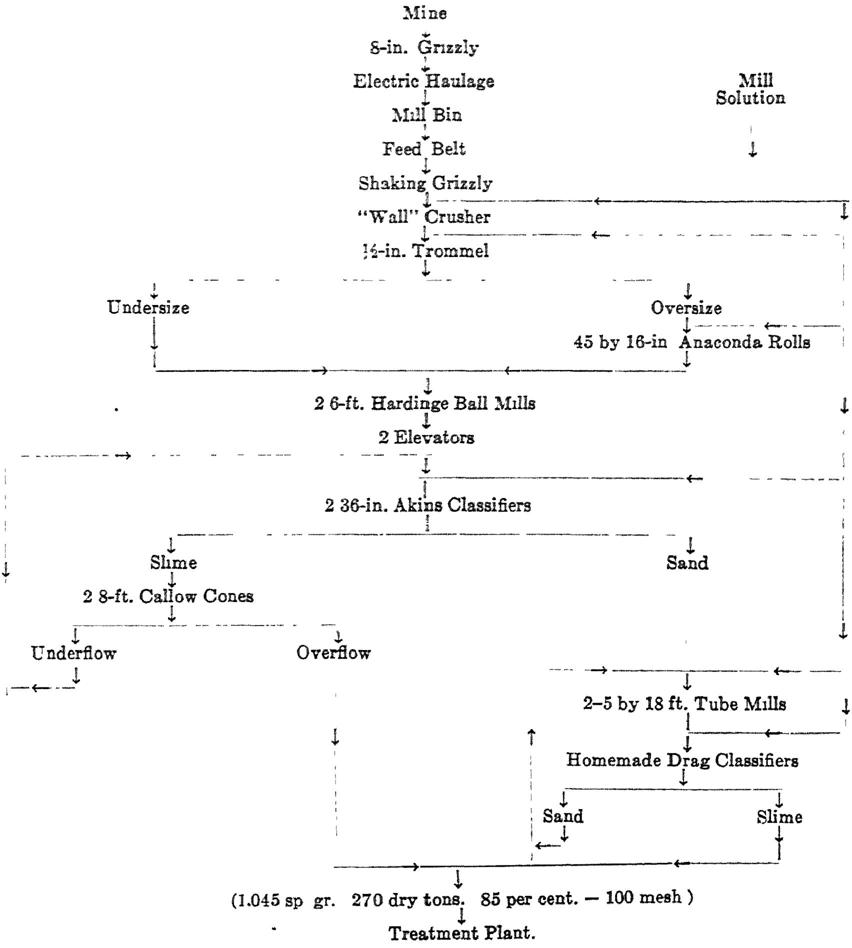
	Per Ton	
MINING		
Ore Breaking:		
Labor	$0.259	
Supplies	0.098	$0 357
Tramming 100-ft. Level:		
Labor	0.044	
Supplies	0.002	0.046
Timbering		0.013
Electric Haulage:		
Labor	0.084	
Supplies	0.006	
Power	0.022	0.112
General Expense:		
Surface drainage	0 004	
Haulage tunnel repairs	0.039	
Assaying and sampling	0.027	
Surveyor	0.011	
Superintendence	0.067	
Incidentals	0.024	
Development	0.083	
Overburden and waste	0 174	0.429
Grand Total		$0.957
MILLING		
Crusher and Rolls (Wall-toothed Roll and 45-in. Roll):	Per Ton	
Labor	$0.057	
Supplies	0.003	
Power	0.034	$0.094
Hardinge Ball Mills (Two 6-ft. Mills):		
Labor	0.050	
Supplies	0.026	
Power	0.076	0.152

Elevating and Separating (Two 36-in. Akins Classifiers):			
Labor	0.006		
Supplies	0.001		
Power	0.002	0.009	
Tube Milling (Two 5-ft. by 18-ft.):			
Labor	0.011		
Supplies	0 049		
Power	0.092	0.152	
Agitation (Three 32-ft. by 4-ft. Dorrs):			
Labor	0.028		
Supplies	0.006		
Power	0.027	0.061	
Chemicals:			
Cyanide	0.218		
Lime	0.293		
Lead acetate	0 024	0.535	
Continuous Decantation (Six 35-ft. by 12-ft. Dorrs):			
Labor	0.004		
Power	0.017	0.021	
Filtering and Discharging (Six 14-ft. by 12-ft. Olivers):			
Labor	0.062		
Supplies	0.045		
Power	0.088	0.195	
Precipitation:			
Labor	0.025		
Supplies	0.067		
Power	0.019	0 111	
Return Pumping:			
Labor	0.009		
Power	0.012	0.021	
Refining:			
Labor	0 023		
Supplies	0.021		
Power	0.001	0.045	
Assaying and Sampling:			
Labor	0.018		
Supplies	0.008	0.026	
Superintendent and Foremen		0.090	
Experiments		0.010	
General Expense		0.043	
Water Supply		0.021	
Grand Total Milling			$1.586
Grand Total Mining			0 957
Grand Total Mining and Milling			$2.55

TABLE 2.—*Milling Data*

	Figures Obtained by Dehydrating Samples		Figures Obtained by Drying Below 110° C		Comparative Figure for Quartz, 2.7 Specific Gravity
	H_2O, Per Cent.	2.4 Specific Gravity, Dry Tons	Moisture, Per Cent.	1.9 Specific Gravity, Dry Tons	
Ore milled per month, 10,000 gross tons	19	8,100	4	9,600	2 per cent H_2O
Ore milled per day, 333 gross tons	19	270	4	320	2 per cent. H_2O
Ball mill discharge, specific gravity, 1.439	48	270	36	320	
Tube mill discharge, specific gravity, 1.394	52	234	40	279	Specific gravity, 1.64; moisture, 38 per cent.
Akins classifier, sand product	38	175	25	208	Moisture, 24 per cent.
Slime to treatment plant, specific gravity, 1.045	1 to 11 5	270	1 to 10	320	1 to 7, specific gravity, 1.08
Primary thickeners underflow, specific gravity, 1.15; settling area, 8 sq. ft. per ton	1 to 3 5	270	1 to 2 57	320	1 to 2, specific gravity, 1.26 or better.
Secondary thickener underflow, specific gravity, 1.23; settling area, 20 sq ft. per ton	1 to 2.1	270	1 to 1.5	320	1 to 1, specific gravity, 1.46 or better.
Filter cake	43	270	30	320	Moisture, 29 per cent.

TABLE 3.—*Flow Sheet.*



Comparisons between Electrolytic Copper and Two Varieties of Arsenical Lake Copper with Respect to Strength and Ductility in Cold-Worked and Annealed Test Strips*

BY C. H. MATHEWSON,† PH. D, NEW HAVEN, CONN., AND E. M. THALHEIMER,§ NEW HAVEN, CONN.

(Arizona Meeting, September, 1916)

CHARACTER OF THE WORK IN HAND

IN planning the present experiments, we have made a particular effort to secure that adjustment of working conditions which would render the forthcoming tests most serviceable by way of indicating the comparative properties of the three different commercial grades of copper used. More specifically, we have endeavored to obtain certain definite information with respect to the relative behavior under test of ordinary electrolytic copper and furnace-refined lake copper containing (*a*) a moderate percentage of arsenic and (*b*) a rather large percentage of arsenic, in addition to the usual content of silver and other impurities.

These comparisons were made systematic and comprehensive by testing throughout a series of (eight) reductions running from 0 to 70 per cent. in the case of cold-rolled metal, and throughout a series of (15) temperatures running from 100° C. to 1,000° C. in the case of annealed metal. In other words, a survey of properties similar to that made by Grard,[1] in the case of electrolytic copper, was here repeated (with modifications) and extended to two other commercial varieties of copper.

The great importance of copper both as a material of construction and as the base of our most important alloys (aside from steel) has led to innumerable investigations of its properties, particularly in commercial forms. Naturally, the merits of furnace-refined lake coppers *vs.* those of the very pure electrolytic product have furnished the subject of much experimental work and animated discussion, wherein the effect of arsenic has by no means been overlooked. In the literature of the past 10 years,

* Metallographic communication from the Hammond Laboratory of the Sheffield Scientific School, Yale University.

† Assistant Professor of Metallurgy, Sheffield Scientific School, Yale University.

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[1] C. Grard: Cuivre et Laitons à Cartouches, *Revue d'Artillerie*, February–April, 1909.

we have found a number of papers in which the ordinary mechanical properties of various brands of copper have been compared after more or less limited or arbitrary forms of treatment, but no paper describing comparative tests after systematic variation of thermal and mechanical treatment. While much of this material is valuable, nevertheless, owing to its diverse and fragmentary character, we have not thought it desirable to use it as a general basis for discussion in connection with our own tests, or, indeed, to attempt any critical review of the literature bearing upon the present subject.

It may be remarked that our experiments do not deal with the effect of arsenic in the absence of other common impurities—little or no progress appears to have been made in this direction since the publication of Bengough and Hill's experiments in 1910[2]—but refer exclusively to comparisons between electrolytic copper and two arsenical brands produced by the Michigan Smelting Co. throughout a wide range of treatment. It is our belief that the interests of consumers and other parties to whom comparative tests are of value may best be served by simple presentation of the results of these tests, leaving special questions of interpretation in the hands of any interested reader. Summaries of recent literature dealing with the properties and structure of copper in various forms may be found in Hofman's *Metallurgy of Copper*[3] and in reports of the Symposium on the Metallurgy of Copper, presented at the meeting of the International Engineering Congress, 1915, in San Francisco.[4]

It may also be appropriate to remark that we have restricted our work to the determination of tensile strength (maximum-load strength), elongation, and reduction of area—properties which are used as a basis for judgment as to quality affecting most engineering requirements. Since there are other criteria of judgment—some specific and capable of routine development, such as the determination of elastic limit, modulus of elasticity and of strength properties under different forms of loading, or at elevated temperatures; others not so well standardized, but, nevertheless, susceptible to quantitative treatment, such as electrochemical potential (rate of solution), varied chemical behavior, etc.; and still others which can only be gaged by manufacturing or service conditions, such as surface qualities, or peculiarities, during drawing operations—we have by no means exhausted the possibilities of comparison in the present series of tests.

Questions of homogeneity with respect to the distribution of oxide

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[2] G. D. Bengough and B. P. Hill: The Properties and Constitution of Copper-Arsenic Alloys, *Journal of the Institute of Metals*, vol. 3, pp. 34–97 (1910).

[3] McGraw-Hill Book Co., New York, 1914.

[4] Papers by William Campbell on the Metallography of Copper and by C. R. Haywood on the Physical Properties of Copper.

and other micrographic features have been dealt with in a limited sense in the present paper, as have questions of resistance to the destructive agency of reducing gases during heat treatment ("gassing" of copper).

It has not been possible to investigate the effect of degree of deformation (reduction by rolling) and of time in coördination with temperature and strength properties, in connection with the annealing experiments, since such an elaboration of the work would necessitate the performance of several thousand instead of several hundred individual tests. A general treatment of this subject has been attempted by one of us[5] and three-dimensional annealing diagrams (degree of deformation, percentage decrease in tensile strength, temperature for definite period of anneal; degree of deformation, percentage decrease in tensile strength, period of anneal at constant temperature), based upon a limited amount of experimental work in the case of "magnesium bronze" (copper, deoxidized with magnesium) have been given by von Müller.[6]

After due consideration of the questions raised above, we decided to use an annealing period of 40 min. and a degree of deformation amounting to a 50 per cent. reduction in area of section by cold-rolling as a basis for all annealing tests. Such procedure is not inconsistent with ordinary annealing practice and cannot fail to develop the principal annealing characteristics of the material in hand.

Composition of the Metal Tested

Metal was sent to the rolling mill in the form of commercial cakes, 14 by 17 by 3 in., weighing in the neighborhood of 275 lb.

Two of these cakes were cast at the works of the Michigan Smelting Co., Houghton, Mich., and these represent the average quality of brands known as Mohawk and Copper Range copper, respectively. The other cake was obtained from The United States Metals Refining Co. ("D R W" copper) and is taken to represent electrolytic copper of average quality.

Determinations of oxygen and arsenic were made in the laboratory of the Michigan Smelting Co. Determinations of silver were made in the Hammond Laboratory.

Details of analytical procedure need not be introduced. All determinations were made by experienced chemists and we have every confidence in their accuracy. No determination was made of the very small quantities of iron, sulphur, nickel, cobalt, tellurium, etc., which may be present in the metal.

[5] C. H. Mathewson: A Metallographic Examination of Some Ancient Bronzes *American Journal of Science*, Series 4, vol. 40, pp. 550–555 (1915).

[6] von Müller: Beitrag zur Erkenntniss des Einflusses der Glühdauer auf die Erweichung verschieden stark gereckter Leitungsbronze, *Metall und Erz*, Neue Folge, vol. 3, pp. 213–220 (1915).

The analytical results follow:

	Oxygen, Per Cent.	Arsenic, Per Cent.	Silver	
			Per Cent.	Ounces
Electrolytic copper	0 071	0.000	0 0005	0.15
Mohawk copper	0.052	0.096	0.069	20.0
Copper Range copper	0 055	0.296	0 052	15.0

Previous History of Test Pieces and Experimental Procedure

The experimental work was divided into two parts: (1) determination of the alteration in strength and ductility (strain hardening) produced by rolling the three kinds of material from an initially soft condition through a series of reductions as specified below, and (2) determination of the alteration in strength and ductility (removal of strain hardening) brought about by annealing the three kinds of material after a previous reduction of 50 per cent. in area of section by rolling.

Accordingly, two separate sets of test strips were prepared. In preparing both sets, the original cake was first planed to a thickness of 2 in., then hot-rolled and finally cold-rolled and annealed before variation in treatment designed to produce the two different sets of test strips was introduced. This procedure insured the production of test strips which had been thoroughly worked, both hot and cold, under conditions which are representative of the treatment commonly received by most mill products of this sort.

Preparation of the First Set of Test Strips

One of two methods may be selected in preparing a set of test strips of different tempers (set 1 according to the above designation), *i.e.*, the strips may be finished at a succession of gages, or all of them may be finished at the same gage.

Steps (*a*), (*b*), and (*c*), of the accompanying diagram, Fig. 1, represent the preliminary stages of rolling which may be considered common to both methods. The relative changes in thickness from one stage of reduction to another are correctly shown in the sketches and the actual gages and percentage reductions between stages are written into the sketches. Sketch (*a*) represents the original cake planed to a thickness of 2 in.; sketch (*b*) represents the material after hot-rolling to gage No. 0, B. & S. (0.325 in.), a reduction of 83.75 per cent.; and sketch (*c*) represents this material after cold-rolling to gage No. 8 (0.128 in.), a reduction of 60.6 per cent. At this point the metal is annealed.

If it is desired that the final product shall be obtained according to the first method, the annealed metal (*c*) is cut into strips which are rolled to a series of measured reductions, in which case the thickness will decrease progressively from that of the first strip (minimum reduction) to that of the last (maximum reduction).

This method was adopted in the present investigation and the numeri-

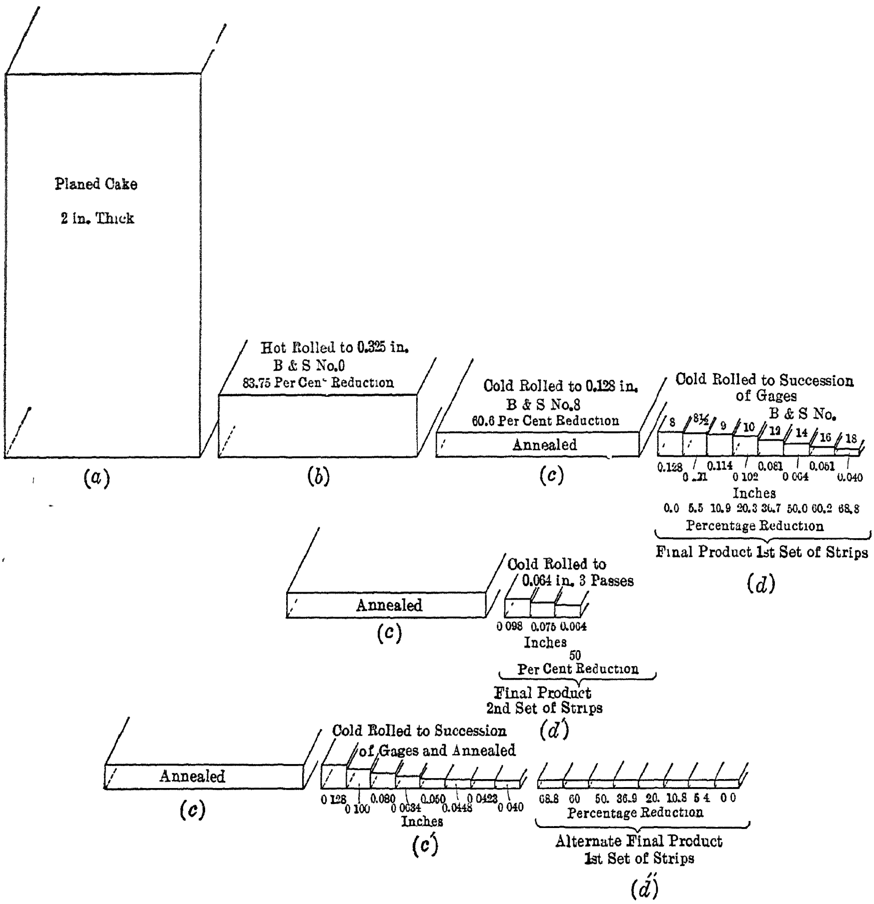


FIG. 1.—DIAGRAM TO REPRESENT PREPARATION OF TEST STRIPS.

cal specifications sent to the mill[7] to cover this part of the work are written into sketch (*d*) of Fig. 1, which represents the series of eight strips composing this set. For convenience, even B. & S. gage numbers running from 8 to 18 (exception, 8½) were used to limit the separate stages. The percentage of the initial thickness (0.128 in. in all cases) taken off in

[7] The material used in all of the tests described in this paper was rolled by the Bridgeport Brass Co.

the reduction is entered below the value for the final thickness under the sketch of each respective strip. Thus, in the case of the last strip:

$$\frac{0.128 \text{ in. (initial thickness)} - 0.040 \text{ (final thickness)}}{0.128 \text{ in. (initial thickness)}} \times 100 = 68.8.$$

In view of the convenient survey of the rolling stages afforded by the sketches of Fig. 1, it appears superfluous to draw up a table of the data involved.

If it is desired that the final product shall be obtained according to the second method (which was not used in these experiments), stages (c') and (d'') may be substituted for stage (d). According to (c'), the metal from (c) is cut into strips which are given a series of reductions as shown, so that, after annealing the lot, another set of reductions will produce test pieces of the same thickness, but different tempers, as indicated in (d''). In order to make the comparison perfectly clear, the reductions marked in sketch (c') have been calculated so as to yield the same series of reductions after the strips have been rolled to a final product of uniform thickness (0.040 in.) as was obtained in the final product according to the first method. The final reductions are marked in sketch (d'').

Since we had to consider carefully the respective advantages of these two methods, it seems appropriate to devote some space to a consideration of this subject. It is unquestionably desirable, other things being equal, to perform all tests upon strips of the same cross-sectional area. The relation between surface and volume of the test piece influences to some extent the testing value calculated per unit of sectional area. There appears to be a "skin effect" which tends to produce higher values of tensile strength in thin specimens than in thick specimens. Notably, in very thin specimens, the testing results are considerably dependent upon the surface conditions of the test piece and comparatively insignificant scratches, or indentations, may lead to premature rupture.[8]

There are, however, certain difficulties in the way of producing a satisfactory set of test strips of uniform thickness and varying percentages of reduction. In the first place, the different samples must receive different amounts of work before the necessary anneal which precedes the final impression of a graded series of reductions bringing all strips to the same finishing thickness. This means that the micrographic characteristics, notably grain size, of the different strips cannot be uniform; those which received a heavy reduction before the anneal will possess a more highly refined grain than those which received a light reduction. It is

[8] Metallographists are generally familiar with the phenomenon of surface flow, first demonstrated by Beilby, which produces a surface hardening, quite marked in the case of soft metals such as copper, during any ordinary process of surface dressing or finishing.

true that we may depart somewhat from the order of changes sketched in (*c*) and (*c'*) of Fig. 1 by cold-rolling in (*c*) to a lesser degree, say, to gage 0.220 in., instead of gage 0.128 in., omitting the anneal at this point, and continuing the rolling so as to produce strips of the gages shown in (*c'*). All of these strips would then have received comparatively heavy reductions and grain refining would be more uniform in the ensuing anneal. This procedure would modify but not entirely remove the difficulty in question.

From a practical standpoint, there would be no check on the rolling-mill work, since the different gages in the intermediate product are all reduced to the same gage in the final product and any departure from specifications in the intermediate work could not be recognized in the final product. This is serious, because such inaccuracies might greatly influence the temper of the product, particularly in the case of light reductions. It may be noted, in this connection, that to produce a final product of gage 0.040 in., having a reduction of 5.4 per cent., the strip is required to be only 0.0423 in. thick before rolling. This requirement is too rigid for ordinary mill work. When the same reduction is made according to the first method, it is performed upon a thicker strip and 0.007 in., instead of 0.0023 in., is taken off. To eliminate this difficulty, the whole set of strips would have to be finished thicker.

In the case of the first method, we are in possession of a reliable check upon the rolling-mill work, since the amount of reduction shows directly in the individual strips and may be measured up with the micrometer.

On the basis of these remarks, it is perfectly clear that the first method is preferable when the metal must be sent to a rolling mill for preparation. With respect to the shortcomings of metal prepared in this way, it may be stated that from the standpoint of comparison between different brands of metal, as in the present case, such very largely disappear, since they only affect the comparison between different tempers and the validity of the succession of testing values thereby secured. It is here assumed that the different kinds of metal follow the same schedule in the mill.

Preparation of the Second Set of Test Strips

The test strips of the second set were finished at gage No. 14 (0.064 in.) with a reduction of 50 per cent., taken off in three passes beginning with annealed metal at gage No. 8 (0.128 in.). These strips were taken from the same cakes as the first set of strips. The rolling operations are, accordingly, those shown in Fig. 1, (*a*), (*b*), and (*c*) followed by (*d'*), instead of (*d*).

Specifications

It has already been stated that the numerical specifications for rolling are indicated in the sketches of Fig. 1. In sending these specifications

to the mill, it was requested that the rolling conform as closely as possible to the data submitted, but a margin of 0.003 in. above or below any given gage was allowed. The mill work was conducted under the supervision of the works laboratory and we were generally satisfied with the product.

Slight departure from the specifications scarcely affects the percentage reductions figured from the specifications except in the case of very light reductions, as in the first two of the first set. Here, the actual reductions varied considerably in the three kinds of material, which would not have been the case if each had been passed through the same setting of the rolls. This was recommended but did not appear feasible in the mill. However, such variations are not serious, since we have the data for calculating the exact reductions obtained. All of the data in question will be found in Tables 2 and 3.

Since we are chiefly interested in comparisons between the three kinds of material, it is important that all receive the same treatment in prescribed annealing operations. On this account, it was insistently urged that the different lots be annealed simultaneously and in the same part of the load. From our micrographic examination, we believe that these specifications, which called for uniform anneal for 1 hr., at 650° to 700° C., were properly carried out.

Procedure with the First Set

In the case of the first set of test strips, the laboratory work consisted solely in testing the prepared strips.

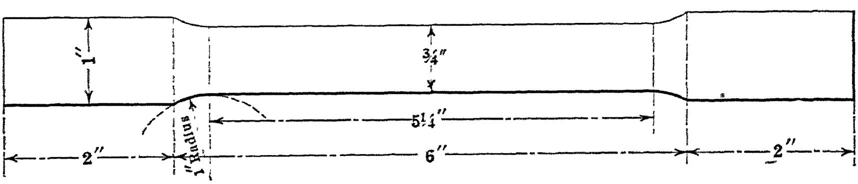


FIG. 2.—DIMENSIONS OF TEST STRIP.

For this purpose, a 50,000-lb. Riehle Universal testing machine was used. Each final result represents the average of tests on four separate strips. A uniform rate of loading was used in all cases, light scratch marks instead of punch marks were made on the strips, and it is believed that all essential precautions were observed in the interest of obtaining the most trustworthy results. Test pieces were accurately milled according to the accompanying sketch (Fig. 2). It may be noted from this sketch that the original strips were 10 in. long by 1 in. wide, while, in milling, three-quarters of the original sectional area was retained in the test section. Our intention in this respect was to cut away just enough material from the sides to insure a properly located break without de-

creasing the breaking load to a point which would render the errors at the machine serious in comparison. The thinnest (most severely reduced) strips (0.040 in.), although the strongest per unit of sectional area, broke under the lightest load, viz., some 1,700 lb. The minimum breaking load for the regular annealed strips of the second set (0.064 in.) was some 1,250 lb., while the weakest of the small number of strips tested after anneal in a reducing atmosphere broke at 930 lb. Thus, if the machine can be read accurately to 10 lb., the maximum testing error is only about 1 per cent.

Breaks were considered faulty when they could not be placed near the middle of a 2-in. length selected with the aid of the scratch marks along the milled section of the strip, viz., elongation was measured in 2 in. and the measurements were made with proper regard for the distribution of elongation along the broken strip.

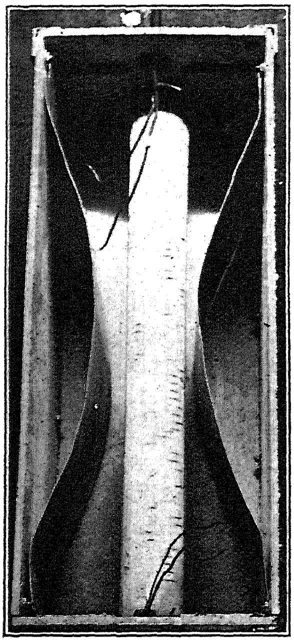


Fig. 3.—Interior of Annealing Furnace with Filling Removed.

Measurements of reduction of area were made with the micrometer but, owing to the comparatively small sectional area of the strips, the results cannot be considered very accurate, particularly in the case of all values below about 25 per cent.

Procedure with the Second Set

The strips of the second set were all tested after a prescribed annealing treatment. The following is a summary of the methods adopted in this part of the work: The annealing was done in a ribbon-wound alundum tube, 18 in. long and 1½ in. in inside diameter. This resistor was covered with alundum cement and mounted in a rectangular box made of transite asbestos wood, as shown in Fig. 3.

In order to flatten the thermal gradient in the tube, the heat capacity of the furnace was increased by means of a loose filling of granular fire clay, and false inner sides bent as indicated in the figure were used to cut down the quantity of filling around the central portion of the tube (which would ordinarily run hottest). In this way, the loss of heat along the sides of the furnace was made to decrease toward the end-regions, thereby compensating for the extra loss at the ends of the furnace.

The gradients along the central 10 in. of the tube, in which the test strips were located, are shown in Fig. 4 at three different temperatures in different parts of the annealing range. It may be noted that the center

of the furnace was kept at a constant temperature (the furnace was allowed to come to equilibrium with a fixed energy input), while the gradients were taken on one side, and again, at a constant temperature in another series of tests (not exactly the same temperature as before), while the gradients were taken on the other side. Interpreted in this sense, these curves show a maximum variation from center to ends (central region of 10 in.) of 15° at approximately 300°; 11° at approximately 600°, and 22° at approximately 900° C. These gradients are made still flatter by the balancing effect of the load.

Four milled strips were annealed at once. It was therefore necessary to make three runs at a given temperature in order to handle the 12 strips

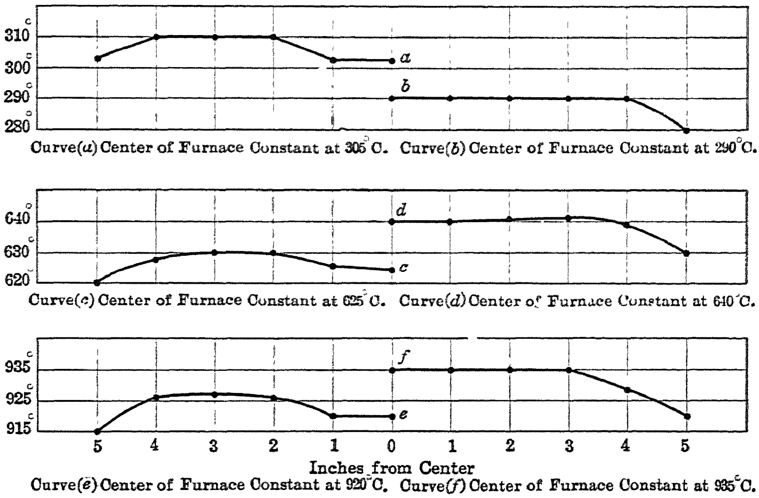


FIG. 4.—TEMPERATURE GRADIENTS OF ANNEALING FURNACE AT APPROXIMATELY 300°, 600° AND 900° C.

required (four each from three kinds of metal). We were able to make these different runs under practically identical conditions, but, in anticipation of possible variations of treatment, each load was composed of strips from the different kinds of metal, so that any differences shown by the average of tests on the several kinds of metal could not be ascribed to differences in annealing treatment. Marking the electrolytic samples 1, the Mohawk samples 2, and the Copper Range samples 3, the different loads were made up as follows:

First Load	Second Load	Third Load
1—2—3—1	2—3—1—2	3—1—2—3

In starting an anneal, the furnace was brought to a temperature some 10° above the actual annealing temperature sought and then the four strips were pushed into the center of the furnace tube side by side with

the end of the thermocouple tube as nearly as possible in the center of the load. When the temperature reached the desired value, attention was paid to the resistances in series with the furnace tube and such alterations as were necessary to keep the temperature constant were made from time to time on the basis of 5- or 10-min. observations of temperature.

The galvanometric method of measuring temperature was employed in connection with an ordinary platinum, platinum-rhodium thermocouple (Siemens and Halske galvanometer of 410 ohms internal resistance). The hot junction of the thermocouple remained in perfect condition throughout all of the experiments. Standardization was made at the freezing points of pure tin, 232°; zinc, 419°; antimony, 631°; and silver (in hydrogen), 961°, keeping the cold junction at 20°. The same conditions were observed in making all anneals. An empirical correction curve of the temperature scale of the galvanometer was made on the basis of the standardization results and all temperatures were corrected by interpolation. Re-standardization of the apparatus after completion of the experiments showed that constancy had been maintained within 10°.

A typical annealing curve shows a preheating period of about 20 min., during which the temperature rises gradually to the value chosen for annealing, and an annealing period of 40 min., during which the temperature varies only a few degrees. Owing to the uniformity which was attained in making these anneals, it does not appear desirable to devote space to an elaborate tabulation of the annealing data. We believe that the annealing temperatures reported are consistently accurate (making allowance for all sources of error) within 15°.

TABLE 1

Temperature of Anneal, Degrees Centigrade	Preheating Period in Minutes			
	Load No. 1	Load No. 2	Load No. 3	Average
100 water	4	2	2	2.7
200 oil	4	4	3	3.7
250 Electric furnace	21	22	20	21.0
275 Electric furnace	20	20	21	20.3
300 Electric furnace	19	16	17	17.3
325 Electric furnace	22	22	23	22.3
350 Electric furnace	19	19	19	19.0
400 Electric furnace	21	18	20	19.7
500 Electric furnace	19	15	14	16.0
600 Electric furnace	18	18	19	18.3
700 Electric furnace	17	18	18	17.7
800 Electric furnace	15	15	15	15.0
900 Electric furnace	12	12	13	12.3
1,000 Electric furnace	13	12	18	14.3

With respect to the preheating periods, which were naturally subject to more or less variation in the different experiments, a summary of the principal data appears to be desirable and this is given in Table 1.

Some of the strips of the second set were annealed in an atmosphere of illuminating (coal) gas. The approximate composition of this gas by volume is as follows:

	Per Cent.
Hydrocarbons	48.5
Hydrogen	25.0
Carbon monoxide	18.0
Carbon dioxide	2.0
Oxygen	0 5
Nitrogen	6.0

These anneals were made in a furnace similar to the one previously described, in which an iron tube was substituted for the alundum tube and both ends were closed except for a small gas inlet at one end and exit at the other end. The thermocouple was introduced at intervals through a suitable hole at one end and left in the furnace long enough to reach the temperature of its surroundings, after which it was withdrawn to avoid contamination by long exposure to the reducing gas, and the hole plugged up. Less accurate control was secured in these experiments and the preheating periods were not determined.

Strips from the different kinds of metal were mixed in each load and, aside from all questions as to the ultimate accuracy of the temperature measurements, valid comparisons between the different kinds of metal (which was the main object of all tests) were obtained.

Samples for micrographic examination (1 by 1 in.) were cut from the ends of the test pieces after annealing.

Tabulation of Results

(1) *Tests in the Cold-Rolled Condition. First Set of Strips*

Table 2 shows the results of tests made on strips rolled to a succession of gages beginning with No. 8 (B. & S.), viz., 0.128 in., and ending with No. 18, viz., 0.040 in. Test strips were milled according to Fig. 2. Micrometer measurements are not given in the table except in case of the thickness. The latter is of importance in indicating the character of the product with respect to temper and an average must be used in calculating the percentage reduction of area effected by rolling, while other measurements are of a routine character, as prerequisites to the calculations of strength and ductility.

The previous history of the strips is given in the sketches of Fig. 1.

TABLE 2.—*Strength and Ductility of Cold-Rolled Strips*

Average thickness before rolling: Electrolytic Copper, 0.1310 in.; Mohawk Copper, 0.1284 in.; Copper Range Copper, 0.1316 in.

No.	Brand of Copper and Reduction by Rolling*	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
1	Electrolytic copper, reduced 0.0 per cent by rolling.	0.1315	33,100	40.5	52.0
2		0.1310	32,900	41.0	52.0
3		0.1310	32,500	41.9	50.0
4		0.1310	32,900	40.6	51.5
	Average	0.1310	32,850	41.0	51.4
5	Mohawk copper reduced 0.0 per cent. by rolling	0.1280	32,700	43.7	51.5
6		0.1285	33,700	44.0	54.5
7		0.1280	33,500	45.5	54.5
8		0.1290	33,100	45.0	54.0
	Average	0.1284	33,250	44.6	53.6
9	Copper Range copper reduced 0.0 per cent by rolling.	0.1320	33,100	43.5	53.5
10		0.1310	32,400	43.1	53.0
11		0.1315	33,100	45.0	53.0
12		0.1320	33,100	45.2	53.0
	Average	0.1316	32,925	44.2	53.1
13	Electrolytic copper reduced 6.9 per cent by rolling.	0.1220	34,800	39.3	46.0
14		0.1220	34,300	38.0	45.0
15		0.1220	34,800	38.7	42.0
16		0.1220	34,700	40.0	44.5
	Average	0.1220	34,650	39.0	44.4
17	Mohawk copper reduced 6.2 per cent. by rolling	0.1200	34,200	44.3	45.0
18		0.1200	34,600	44.0	44.0
19		0.1205	35,200	44.1	42.5
20		0.1210	34,100	44.5	45.0
	Average	0.1204	34,525	44.2	44.1
21	Copper Range copper reduced 8.7 per cent by rolling.	0.1200	35,500	39.5	38.0
22		0.1205	35,600	36.2	34.0
23		0.1200	35,100	40.6	38.0
24		0.1200	35,600	39.3	38.0
	Average	0.1201	35,450	38.9	37.0
25	Electrolytic copper reduced 12.9 per cent by rolling.	0.1140	37,300	35.9	34.0
26		0.1140	36,900	36.2	35.5
27		0.1140	36,800	35.8	34.0
28		0.1145	36,100	35.8	34.0
	Average	0.1141	36,775	35.9	34.4
29	Mohawk copper reduced 11.1 per cent. by rolling.	0.1140	36,800	40.2	36.0
30		0.1145	36,600	40.7	36.5
31		0.1140	37,000	40.0	38.0
32		0.1145	37,100	39.8	38.0
	Average	0.1142	36,875	40.2	37.1
33	Copper Range copper reduced 13.1 per cent. by rolling.	0.1145	38,100	33.7	31.0
34		0.1145	38,300	33.7	29.0
35		0.1140	36,900	32.2	27.5
36		0.1145	37,500	32.3	31.0
	Average	0.1144	37,700	33.0	29.6
37	Electrolytic copper reduced 21.4 per cent. by rolling.	0.1030	40,900	23.4	13.5
38		0.1030	41,400	22.5	12.5
39		0.1030	41,000	22.5	9.5
40		0.1030	41,000	24.4	16.5
	Average	0.1030	41,075	23.2	13.0
41	Mohawk copper reduced 20.2 per cent. by rolling.	0.1025	41,600	27.3	16.0
42		0.1025	40,000	28.1	14.0
43		0.1020	40,600	29.8	16.0
44		0.1025	40,500	29.8	16.0
	Average	0.1024	40,675	28.7	15.5
45	Copper Range copper reduced 22.5 per cent. by rolling.	0.1020	43,700	22.9	10.5
46		0.1020	42,900	21.9	10.0
47		0.1020	42,800	22.5	9.5
48		0.1020	43,000	23.5	11.0
	Average	0.1020	43,100	22.7	10.3

* Calculated from average thickness of strips and average thickness before rolling.

TABLE 2.—*Strength and Ductility of Cold-Rolled Strips (Continued)*

Average thickness before rolling: Electrolytic Copper, 0.1310 in.; Mohawk Copper, 0.1284 in.; Copper Range Copper, 0.1316 in.

No.	Brand of Copper and Reduction by Rolling*	Thickness, Inches	Tensile Strength in Pounds per Sq In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
49	Electrolytic copper re-	0.0810	47,900	10.0	5.0
50	duced 38.2 per cent	0.0810	47,700	11.0	5.0
51	by rolling.	0.0810	47,900	7.5	5.0
52		0.0810	47,500	8.7	5.0
	Average...........	0.0810	47,750	9.3	5.0
53	Mohawk copper reduced	0.0810	49,100	11.5	6.0
54	37.2 per cent. by	0.0805	49,500	10.2	5.0
55	rolling.	0.0810	49,600	9.4	5.0
56		0.0805	49,600	9.4	5.0
	Average...........	0.0807	49,450	10.1	5.2
57	Copper Range copper	0.0815	50,500	12.1	5.0
58	reduced 38.1 per cent.	0.0815	49,800	7.2	5.0
59	by rolling.	0.0815	50,400	7.3	4.0
60		0.0815	50,500	6.2	5.0
	Average	0.0815	50,300	8.2	4.8
61	Electrolytic copper re-	0.0645	49,900	4.2	3.0
62	duced 50.8 per cent.	0.0645	52,400	5.0	3.0
63	by rolling.	0.0645	52,200	5.9	3.5
64		0.0645	53,100	4.9	3.5
	Average	0.0645	51,900	5.0	3.2
65	Mohawk copper reduced	0.0645	53,800	3.0	3.5
66	49.8 per cent. by	0.0645	52,400	4.4	3.0
67	rolling.	0.0645	53,500	5.0	3.0
68		0.0645	52,600	5.2	3.0
	Average...........	0.0645	53 325	4.4	3.1
69	Copper Range copper	0.0640	54,700	2.8	4.0
70	reduced 51.4 per cent.	0.0640	54,000	3.6	4.0
71	by rolling.	0.0640	54,900	4.3	4.0
72		0.0640	55,200	4.3	4.0
	Average	0.0640	54,700	3.7	4.0
73	Electrolytic copper re-	0.0510	54,900	3.9	3.5
74	duced 60.8 per cent	0.0520	55,000	2.4	2.5
75	by rolling.	0.0510	51,000	4.8	2.5
76		0.0515	56,400	3.2	3.5
	Average.........	0.0514	54,325	3.6	3.0
77	Mohawk copper reduced	0.0505	57,000	2.1	3.0
78	60.7 per cent. by	0.0505	57,400	2.1	2.5
79	rolling.	0.0505	57,000	2.1	3.0
80		0.0505	56,800	2.1	3.0
	Average...........	0.0505	57,050	2.1	2.9
81	Copper Range copper	0.0510	57,600	3.9	3.0
82	reduced 61.2 per cent.	0.0510	58,100	3.9	3.0
83	by rolling.	0.0510	57,900	3.4	3.0
84		0.0510	57,400	4.0	3.0
	Average	0.0510	57,750	3.8	3.0
85	Electrolytic copper re-	0.0410	57,000	3.3	3.0
86	duced 69.1 per cent.	0.0400	57,400	3.7	3.5
87	by rolling.	0.0405	57,700	2.3	3.0
	Average...........	0.0405	57,367	3.1	3.2
88	Mohawk copper reduced	0.0405	57,800	2.6	2.0
89	68.1 per cent. by	0.0410	58,400	2.6	2.0
90	rolling.	0.0410	59,000	2.6	2.0
91		0.0410	59,000	2.6	2.0
	Average...........	0.0409	58,550	2.6	2.0
92	Copper Range copper	0.0395	58,800	2.1	1.5
93	reduced 70.0 per cent.	0.0395	59,300	1.4	2.0
94	by rolling.	0.0395	60,500	3.7	2.5
95		0.0395	61,200	2.4	2.5
96		0.0395	60,500	2.4	2.0
	Average...........	0.0395	60,060	2.4	2.1

[1] Calculated from average thickness of strips and average thickness before rolling.

Table 3.—*Strength and Ductility of Strips Annealed* 40 *Min. at Different Temperatures after a Reduction of* 50 *Per Cent. by Rolling**

Average thickness before annealing, 0.064 to 0.067 in.

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
97	Electrolytic copper,	0.0640	51,600	4.56	4.0
98	unannealed (as re-	0.0645	51,600	3.20	4.0
99	ceived from the mill).	0.0670	51,600	4.57	4.0
100		0.0640	51,600	4.56	4.0
	Average........ ...	0.0649	51,600	4.22	4.0
101	Mohawk copper, unan-	0.0670	52,200	4.94	4.0
102	nealed (as received	0.0665	52,100	5.03	4.0
103	from the mill).	0.0665	52,100	6.02	4.0
104		0.0660	52,100	4.66	4.0
	Average...........	0.0665	52,125	5.16	4.0
105	Copper Range copper,	0.0665	53,000	8.83	4.0
106	unannealed (as re-	0.0665	53,500	9.00	4.5
107	ceived from the mill).	0.0665	53,000	8.83	4.5
108		0.0665	52,900	8.60	4.5
	Average...........	0.0665	53,100	8.81	4.4
109	Electrolytic copper an-	0.0640	51,700	3.74	4.0
110	nealed at 100°C.	0.0640	51,600	4.56	4.0
111		0.0640	51,400	4.56	4.0
112		0.0640	51,400	3.96	4.0
	Average...........	0.0640	51,525	4.21	4.0
113	Mohawk copper, an-	0.0660	52,300	4.63	4.0
114	nealed at 100°C.	0.0655	52,900	6.86	5.0
115		0.0660	52,400	5.01	4.0
116		0.0660	52,300	4.63	4.0
	Average...	0.0659	52,475	5.28	4.2
117	Copper Range copper,	0.0660	52,500	8.85	6.0
118	annealed at 100°C.	0.0660	52,500	8.85	6.0
119		0.0660	53,300	7.65	4.5
120		0.0640	54,400	4.77	4.0
	Average...........	0.0655	53,175	7.53	5.1
121	Electrolytic copper, an-	0.0650	49,000	9.86	4.5
122	nealed at 200°C.	0.0645	49,100	9.49	4.5
123		0.0650	48,600	6.12	4.0
124		0.0645	50,500	5.15	6.0
	Average...........	0.0647	49,300	7.66	4.8
125	Mohawk copper, an-	0.0660	52,900	9.45	6.5
126	nealed at 200°C.	0.0660	52,000	8.85	5.0
127		0.0660	52,100	7.45	5.5
128		0.0660	52,000	7.45	5.5
	Average...........	0.0660	52,250	8.30	5.6
129	Copper Range copper,	0.0640	53,100	5.81	5.5
130	annealed at 200°C.	0.0655	52,500	12.95	4.5
131		0.0650	52,700	12.85	4.5
132		0.0660	51,800	13.13	4.5
	Average..........	0.0651	52,525	11.18	4.8

* Calculated values of reduction vary between 49 and 51 per cent.

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
133	Electrolytic copper, an-	0.0645	49,200	20 80	9.0
134	nealed at 250°C.	0.0645	49,100	19.30	9.0
135		0 0650	48,500	19 05	8.5
136		0 0645	48,900	18 30	9 0
	Average. ..	0 0646	48,925	19 36	8.9
137	Mohawk copper, an-	0.0657	51,800	15 30	7.5
138	nealed at 250°C.	0.0650	51,800	14 85	7.5
139		0.0655	51,700	14.00	7.5
140		0.0652	51,700	14 25	8.0
	Average.	0.0654	51,750	14.60	7.6
141	Copper Range copper,	0.0655	52,500	12 00	6.0
142	annealed at 250°C.	0.0660	51,500	14.15	8.0
143		0.0660	53,000	14.15	8.0
144		0.0660	52,600	14.45	7.5
	Average...........	0.0659	52,400	13.69	7.4
145	Electrolytic copper, an-	0.0643	43,000	24.40	19.0
146	nealed at 275°C.	0 0650	43,000	24.60	19.0
147		0.0635	43,600	22.80	(14.0)
148		0.0650	42,500	24.60	20.0
	Average...........	0.0645	43,025	24.10	19.3
149	Mohawk copper, an-	0 0650	51,200	16 65	8 5
150	nealed at 275°C.	0.0658	50,700	15.65	8.5
151		0.0655	50,700	15.40	7.0
152		0 0657	50,700	16 00	8.5
	Average...........	0 0655	50,825	15.92	8.1
153	Copper Range copper	0 0653	52,200	(9.45)	(5.0)
154	annealed at 275°C.	0.0655	51,500	11 00	9.0
155		0.0655	51,000	11 00	9.0
156		0.0655	51,200	11.00	9.0
	Average.....	0.0654	51,475	11.00	9.0
157	Electrolytic copper, an-	0 0640	38,800	36 90	36.5
158	nealed at 300°C.	0.0640	37,600	40.20	36 0
159		0.0650	35,400	(44.00)	(52.5)
160		0.0640	37,700	38.00	37.5
	Average...........	0.0647	37,375	38.37	36.7
161	Mohawk copper, an-	0.0655	50,700	10 75	11.0
162	nealed at 300°C.	0 0660	50,400	15.65	11.0
163		0.0660	50,000	(8.85)	11.0
164		0.0660	50,100	14.10	11.0
	Average...........	0.0659	50,300	13.50	11.0
165	Copper Range copper,	0.0660	50,200	14.50	9.5
166	annealed at 300°C.	0.0645	49,800	13.40	11.5
167		0 0660	51,000	14.35	10.0
168		0.0655	50,400	15.00	9.0
	Average...........	0.0655	50,350	14.31	10.0

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
169	Electrolytic copper, an-	0 0650	34,600	44 60	46 0
170	nealed at 325°C.	0 0650	35,100	43.90	46 0
171		0.0630	34,600	42 00	49 0
172		0.0650	35,600	43 10	(43 0)
173		0.0645	33,800	44.60	49 5
	Average	0.0645	34,740	43 64	47.6
174	Mohawk copper, an-	0.0650	43,000	27.90	19 0
175	nealed at 325°C.	0.0645	43,500	28 40	19.0
176		0 0684	43,500	30.70	18.0
177		0.0650	41,800	29 60	22.0
	Average	0 0657	42,950	29.15	19 5
178	Copper Range copper,	0 0655	37,600	39 90	42.0
179	annealed at 325°C.	0 0655	37,550	39 80	41.0
180		0 0655	36,800	40 20	43.0
181		0 0640	38,200	39.10	42 0
	Average	0 0651	37,540	39 75	42 0
182	Electrolytic copper, an-	0.0650	33,900	45.90	50.0
183	nealed at 350°C.	0.0630	34,300	45.00	51.0
184		0.0645	34,100	46.30	52.0
185		0.0650	33,900	46.40	51.0
	Average	0.0644	34,050	45.90	51.0
186	Mohawk copper, an-	0.0663	34,500	44 00	54.0
187	nealed at 350°C.	0.0650	34,200	45 40	54.0
188		0.0655	34,700	45 40	53.0
189		0 0655	34,700	43 30	54.0
	Average	0.0656	34,525	44 53	53 8
190	Copper Range copper,	0.0650	34,900	43 00	54 0
191	annealed at 350°C.	0 0655	34,700	42 60	54.0
192		0.0660	35,100	43.00	54.0
193		0.0655	34,500	42.20	53.5
	Average..........	0.0655	34,800	42.70	53.9
194	Electrolytic copper, an-	0.0640	34,100	44 60	51.0
195	nealed at 400°C.	0.0645	34,000	45.10	55.0
196		0.0645	33,900	45 00	54.0
197		0.0645	33,000	45 00	51.5
	Average...........	0.0644	33,750	44 93	52.9
198	Mohawk copper, an-	0.0660	34,400	44.50	53.5
199	nealed at 400°C.	0.0660	34,400	45.00	53.5
200		0.0660	34,100	45.50	53.5
201		0.0660	34,500	44.90	54.0
	Average	0.0660	34,350	44.98	53.6
202	Copper Range copper,	0 0655	34,700	47 60	54.0
203	annealed at 400°C.	0 0655	34,600	46 50	53.5
204		0.0645	34,200	47.10	54.0
205		0.0657	34,400	46.10	54.5
	Average..........	0.0650	34,475	46.83	54.0

TABLE 3.—*Strength and Ductility of Strips Annealed* 40 *Min. at Different Temperatures after a Reduction of* 50 *Per Cent. by Rolling (Continued)*

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
206	Electrolytic copper, an-	0.0635	33,500	45.10	57.5
207	nealed at 500°C.	0 0635	33,500	46.50	56.0
208		0 0635	33,200	44.50	49.0
209		0 0640	33,500	46.00	57.0
	Average...........	0.0636	33,425	45.53	54.9
210	Mohawk copper, an-	0.0640	33,200	46.80	49.5
211	nealed at 500°C.	0.0655	33,900	49.30	57 0
212		0 0660	33,500	49.50	56.5
213		0 0650	32,800	49.60	55 0
	Average...........	0 0651	33,350	48 80	54.8
214	Copper Range copper,	0.0655	33,500	47.90	55.0
215	annealed at 500°C.	0.0645	33,400	49.00	58.0
216		0.0645	33,700	45.50	54.5
217		0.0645	33,500	48.50	58.0
	Average..	0.0648	33,525	47.73	56.4
218	Electrolytic copper, an-	0.0645	33,400	46 90	54.5
219	nealed at 600°C.	0.0640	33,300	46.10	54.5
220		0.0640	33,300	46.40	55.0
221		0 0620	33,800	45.40	55.0
	Average...........	0 0636	33,450	46 20	54.8
222	Mohawk copper, an-	0.0645	33,209	48.40	55.5
223	nealed at 600°C	0.0645	33,100	47.00	54.0
224		0.0645	32,900	49 60	56.0
225		0.0655	33,000	49.40	56.5
	Average	0.0648	33,050	48.60	55.5
226	Copper Range copper,	0.0655	33,100	49.40	57.0
227	annealed at 600°C.	0.0655	33,000	48.60	57.0
228		0.0660	33,250	51.50	57.5
229		0.0655	33,200	51.10	56.0
	Average...........	0.0656	33,140	50.15	56.9
230	Electrolytic copper, an-	0.0635	32,200	44.70	52.5
231	nealed at 700°C.	0.0630	31,800	44.90	55.5
232		0.0640	32,199	45.00	55.0
233		0.0635	32,400	46.40	55.0
	Average..........	0.0635	32,125	45.25	54.5
234	Mohawk copper, an-	0.0645	31,900	49.50	54.0
235	nealed at 700°C.	0.0647	32,200	47.90	61.0
236		0.0645	32,200	49.30	55.0
237		0.0645	32,100	49.20	53.5
	Average...........	0.0646	32,050	48.98	55.9
238	Copper Range copper,	0.0650	32,300	50.60	55.5
239	annealed at 700°C.	0.0645	32,100	50.10	59.0
240		0.0643	32,000	49.70	55.0
241		0.0645	31,900	49.70	57.0
	Average...........	0.0646	32,075	50.03	56.6

TABLE 3.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures after a Reduction of 50 Per Cent. by Rolling (Continued)*

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
242	Electrolytic copper, annealed at 800°C.	0 0615	32,400	44 00	49 0
243		0 0615	32,200	44 30	51 0
244		0 0600	32,100	42 80	51.0
245		0 0615	32,500	44 00	50.5
	Average.	0 0615	32,300	43.78	50 4
246	Mohawk copper, annealed at 800°C.	0.0615	32,100	44.80	52 0
247		0 0620	32,200	45 70	53 0
248		0.0630	31,800	45 80	53 0
249		0 0630	31,400	46 30	51 0
	Average.	0 0624	31,875	45 65	52 3
250	Copper Range copper, annealed at 800°C.	0 0620	32,200	46 60	55 0
251		0 0625	32,100	46.00	55.0
252		0 0625	32,100	47.40	59.0
253		0 0613	31,400	47.10	50.0
	Average.	0 0621	31,950	46.79	54.8
254	Electrolytic copper, annealed at 900°C.	0 0600	31,600	39.80	44 0
255		0.0615	31,200	41.40	44.0
256		0 0620	32,200	41.10	46.0
257		0.0600	31,800	39.80	44 0
	Average.	0.0609	31,700	40.53	44 5
258	Mohawk copper, annealed at 900°C.	0.0640	30,100	46.90	47.0
259		0.0620	30,600	42.40	45 0
260		0 0605	30,600	44.60	46 0
261		0.0620	31,500	45.80	44.0
	Average.	0.0621	30,700	44 93	45.5
262	Copper Range copper, annealed at 900°C.	0.0625	30,200	48.50	54.5
263		0.0620	30,200	47.50	51.0
264		0 0610	30,500	45.60	45.0
265		0.0610	30,900	46.50	48.0
	Average.	0 0616	30,450	47.03	49.6
266	Electrolytic copper, annealed at 1,000°C.	0.0600	29,700	37.16	35.5
267		0.0600	30,500	32 10	31.0
268		0.0565	30,900	33.30	30.0
269		0 0570	32,300	31.90	42.5
	Average.	0.0589	30,850	33.62	34 8
270	Mohawk copper, annealed at 1,000°C.	0.0600	30,400	40.40	43.0
271		0 0600	29,800	40.40	38.0
272		0.0605	29,900	40.00	37.0
273		0.0590	30,200	39.40	36.0
	Average.	0.0599	30,075	40.05	38.5
274	Copper Range copper, annealed at 1,000°C.	0.0610	28,800	40.90	35.0
275		0.0580	29,700	38.82	37.0
276		0.0585	29,100	38.60	36.0
277		0.0605	27,600	41.50	37.0
	Average.	0.0595	28,800	39.96	36.3

(2) *Tests after Anneal under Oxidizing Conditions. Second Set of Strips.*

Table 3 shows the results of tests made on strips annealed 40 min. at temperatures ranging from 100° to 1,000°C. The previous history of the strips is given in the sketches of Fig. 1. The methods of annealing are described in the preceding section. Preheating periods are given in Table 1.

While all strips were annealed at gage No. 14 (B. & S.), the thickness is somewhat reduced by oxidation after anneal at high temperatures and, in order that comparisons of this sort may be made, micrometer measurements of thickness are given in the table.

(3) *Tests after Anneal under Reducing Conditions. Second Set of Strips*

Table 4 shows the results of a few tests made on strips annealed 40 min. at 600°, 800°, and 1,000° C., respectively, in an atmosphere of illumi-

TABLE 4.—*Strength and Ductility of Strips Annealed 40 Min. at Different Temperatures in an Atmosphere of Coal Gas**

Average thickness, before annealing, 0.064 to 0.067 in.

No.	Brand of Copper and Annealing Temperature	Thickness, Inches	Tensile Strength in Pounds per Sq. In.	Percentage Reduction of Area	Percentage Elongation in 2 In.
278	Electrolytic copper, an-	0 0690	29,400	Broke at shoulder	(22.0)
279	nealed at 600° C.	0.0680	29,100	Broke at shoulder	(22.0)
	Average..........	0.0685	29,250	....	(22.0)
280	Mohawk copper, an-	0.0690	29,500	Broke at shoulder	(30.0)
281	nealed at 600° C.	0.0670	30,500	Broke at shoulder	(25.5)
	Average...........	0.0680	30,000	....	(27.8)
282	Copper Range copper,	0 0670	31,800	39.20 Good break	41.0
283	annealed at 600° C	0.0670	31,700	39.40 Good break	43.0
	Average...........	0.0670	31,750	39.30	42.0
284	Electrolytic copper, an-	0.0710	21,500	12.35 Good break	12.0
285	nealed at 800° C.	0.0710	17,700	Broke in several	(9.0)
	Average...........	0.0710	19,600	pieces	(10.5)
286	Mohawk copper, an-	0.0700	17,600	Broke at shoulder	(8.0)
287	nealed at 800° C.	0.0690	19,000	Broke in jaws	....
	Average...........	0.0695	18,300		
288	Copper Range copper,	0.0695	20,700	12.80 Good break	11.0
289	annealed at 800° C.	0.0700	21,700	Broke at shoulder	(11 0)
	Average...........	0.0698	21,200	....	(11.0)
290	Electrolytic copper, an-	0.0670	21,900	Broke at shoulder	(14.5)
291	nealed at 1,000° C.	0.0710	22,300	Broke at shoulder	(14.5)
	Average...........	0.0690	22,100	....	(14.5)
292	Mohawk copper, an-	0.0690	21,300	Broke in pieces	(16.5)
293	nealed at 1,000° C.	0.0700	21,700	18.15 Good break	17.0
	Average..........	0.0695	21,500	....	(16.8)
					.....
294	Copper Range copper,	0.0690	22,100	Broke in jaws	
295	annealed at 1,000° C.	0.0690	22,100	16.70 Good break	16.0
	Average...........	0.0690	22,100	....	.... .

* Calculated values of reduction vary between 49 and 51 per cent.

nating gas. The previous history and dimensions of the strips are the same as in the case of the preceding tests.

Discussion of Tables, Curves, and Microstructure

(1) *Tests after Cold-Rolling*

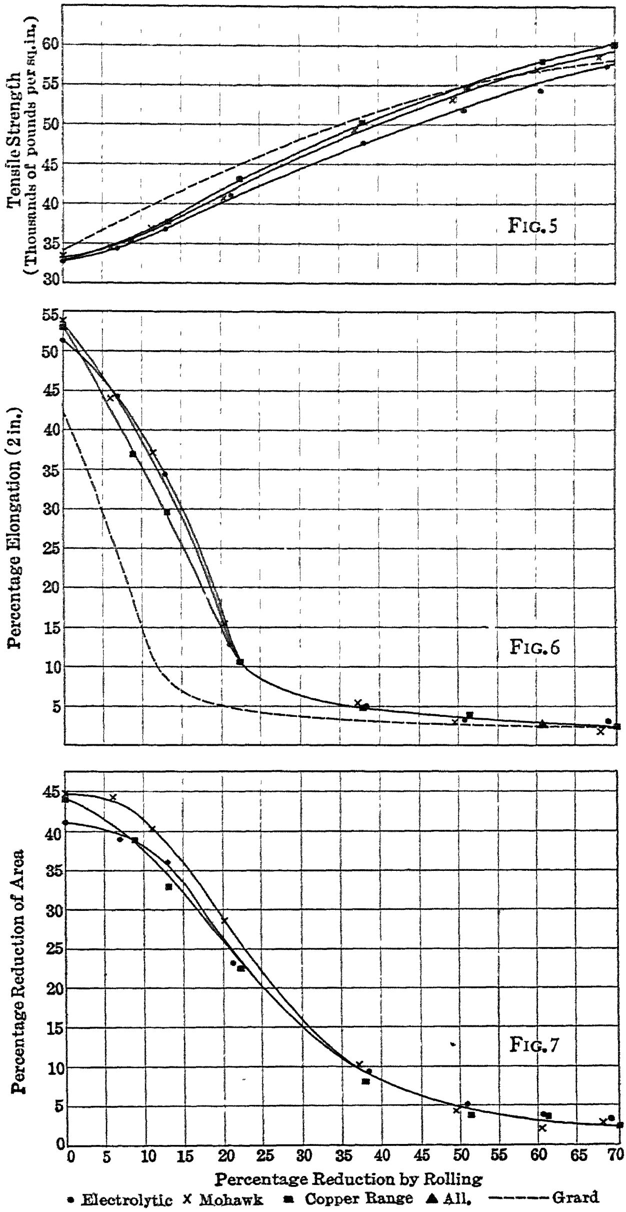
The results assembled in Table 2 have been plotted in Figs. 5, 6, and 7. In Fig. 5, the comparative values of tensile strength in the three kinds of metal are shown; in Fig. 6, the comparative values of elongation; and in Fig. 7, the comparative values of reduction of area.

It will be observed that changes in temper of about the same degree are produced by rolling the three kinds of metal through any given value of reduction. In other words, these tests fail to develop any striking differences of rolling temper between the different brands of copper used. The difficulty of obtaining a high degree of accuracy in the ordinary tensile test is a handicap in comparing products which are very similar in nature. Testing discrepancies of several per cent. will ordinarily occur with careful handling of test strips from the same sheet of metal. The uneven distribution of small percentages of impurity in commercial products is a source of irregularities in this respect. As may be seen in the micrographs (Fig. *d*, Plate 4, is typical of this condition), the oxide particles, originally more or less segregated in the cast cake, are drawn out into continuous trains in the rolled product and it is obvious that there will be a closer grouping of the zones, or attenuated lines of weakness in some cross-sectional areas than in others. Furthermore, the presence of oxide interferes with the normal growth of grain on annealing, and, consequently, there will be irregularities of grain size in the copper matrix, corresponding to segregations of oxide. Such a condition is shown admirably in Fig. *a*, Plate 4, where coarser grain has formed on both sides of an extended zone or band sprinkled with oxide particles.[9] Such conditions cannot fail to be responsible for variations in breaking load, elongation, and reduction of area among the duplicate test strips.

An accurate average (representing the mean properties of the metal) could, of course, be struck from tests on a large number of strips. There is always a limit in this direction, however, and we decided upon four strips as the largest number which could be handled in this investigation. It will be noted from examination of the tabulated results that considerable variations both of strength and elongation were obtained in a given set of tests. It may be left to the individual reader to form his own opinion as to the actual value of the averages reported.

[9] Mathewson and Caesar, in a paper lately sent to the *Internationale Zeitschrift für Metallographie*, have discussed the effect of oxide on the size of recrystallized grain in pure copper.

In rejecting any seemingly abnormal result, we endeavored to be perfectly fair to each kind of metal. In general, where serious irregulari-



FIGS. 5, 6, AND 7.—STRENGTH AND DUCTILITY OF ROLLED STRIPS.

ties developed, an entirely new set of bars was tested and all of the earlier results rejected; in no case were four especially favorable or unfavorable

results selected from a long series of tests, but, in some cases, averages were struck from three tests when there seemed good reason to reject the fourth.

Such differences between the three varieties of copper as appear to be real, although small, may be summarized in the following statements:

The Copper Range product decreases in ductility as measured by elongation more rapidly than the electrolytic product during the early stages of rolling (down to about 20 per cent. reduction of area by rolling), but both products increase in strength about equally, while, during the later stages of rolling, both products decrease about equally in ductility and the Copper Range metal increases in strength more rapidly than the electrolytic metal. The Mohawk copper occupies a more or less intermediate position.

Differences in ductility as measured by reduction of area are also confined to the earlier range of reductions by rolling. While, in principle, the measurement of reduction of area possesses some advantages over the measurement of elongation as a criterion of deformational properties, since the former is more nearly independent of the shape of the test piece, it is far more difficult to make in the case of thin flat metal. Our measurements of elongation are, therefore, more reliable than the corresponding measurements of reduction of area. In the present case, all three kinds of material show practically the same values of reduction of area after the reduction by rolling has passed 35 per cent.

From the general point of view that the Copper Range product takes a harder temper on rolling than the electrolytic product, but does not ultimately become less ductile, it would seem that both might be able to stand the same amount of punishment by rolling, but that the latter might roll a little easier. Since the pressure required for rolling cannot be calculated from the results of the ordinary tensile test, and since it would require, in any event, comparative determinations of the yield-point stress intensities at intervals throughout the rolling range in order to determine which of the two materials would require the greatest load to start permanent deformation at any stage, the above statement must remain subject to eventual qualification by actual test.

Where it is desired to finish a previously annealed product with a limited amount of ductility, but increased strength, by rolling through several numbers of gage, it is quite evident that an appreciable increase in strength without sacrifice of ductility may be secured by using Copper Range metal instead of electrolytic metal. Thus, rolling from No. 8 to No. 12, B. & S. gage, a reduction of about 40 per cent. would give the Copper Range material a tensile strength about 1 ton (per square inch) greater than that of the electrolytic material similarly treated, while the elongation in both cases would be in the neighborhood of 5 per cent.

In order that the well-known curves of Grard[10] (electrolytic copper) may be compared with our own, they have been dotted into Figs. 5 and 6. Grard's measurements of elongation were made on a test section 100 mm. long (approximately 4 in.) and, for this reason, show much lower values than our own, measured in 2 in., until the region of slight ductility is reached at about a 35 per cent. reduction by rolling. From this point on, the stretch becomes sensibly uniform along the entire test length, making it of little consequence whether a length of 2 or 4 in. is chosen, and our results are practically coincident with Grard's.

Grard's curve of breaking strength starts somewhat above and ends somewhat below our curves. Thus, our curves show a higher rate of increase of strength with the degree of reduction by rolling. This is doubtless due to differences in the dimensions of the test strips. For reasons previously given, we used test strips whose thickness decreased progressively with the increase in reduction by rolling (minimum ratio of test length to $\sqrt{\text{sectional area}}$, 6½ to 1; maximum ratio, 11½ to 1), while Grard's practice in this respect is not clear. He shows a ratio of 5 to 1 in a sketch of the test piece, but later states that a variable final cross-section was used.

(2) *Tests after Annealing under Oxidizing Conditions*

The results assembled in Table 3 have been plotted in Figs. 8 (strength), 9 (elongation), and 10 (reduction of area). For general purposes of comparison, Grard's[11] curves from tests on 2-mm. strips of electrolytic copper and Bardwell's[12] curves from tests on No. 12, B. & S. wire, containing from 0.036 to 0.070 per cent. of oxygen and an indeterminate amount of silver, are shown in dotted form.

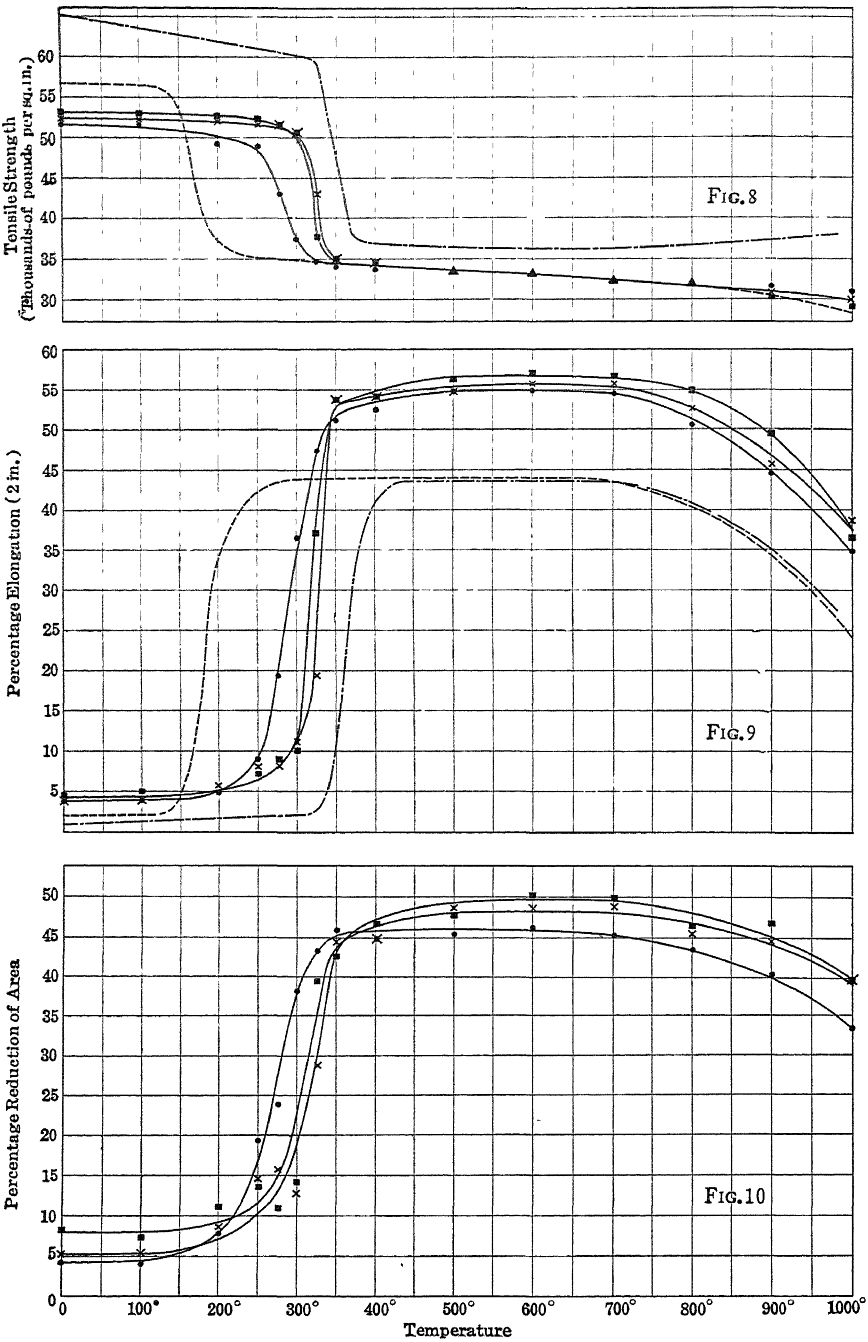
Naturally, one's attention is first drawn to the difference between electrolytic copper, on the one hand, and the arsenical coppers, on the other hand, with respect to the earliest manifestation of an annealing effect. The temperature range of early transformation (annealing period of 40 min.) from the strain-hardened condition to the recrystallized, or annealed, condition is equally narrow in all three kinds of material; it is practically coincident in the two arsenical coppers, but lies some 50° lower in the electrolytic copper.

It is believed that these are the first systematic tests designed to reveal the relative softening (annealing) characteristics of commercially pure copper and commercial copper containing arsenic. The general tendency of arsenic to raise the effective annealing temperature has long

[10] C. Grard: Cuivre et Laitons à Cartouches, *Revue d'Artillerie,* February–April, 1909.

[11] *Loc. cit.*

[12] E. S. Bardwell: The Annealing of Cold-Rolled Copper, *Trans.,* vol. 49, pp. 753 773 (1914).



FIGS. 8, 9, AND 10.—STRENGTH AND DUCTILITY OF ANNEALED STRIPS.

been known on the basis of fragmentary tests, here and there, under a variety of conditions difficult to bring into any sort of alignment. Thus, F. Johnson[13] quotes from Tomlinson to the effect that samples of rolled arsenical copper containing 0.36 per cent. of arsenic did not soften after 3 or 4 weeks at a temperature approaching 400° F. (205° C.), while samples of pure copper were practically annealed. Further, the arsenical copper lost only 6 per cent. of its strength after 4 hr. at 870° C., while the pure copper lost 44 per cent. (*i.e.*, was presumably completely annealed).

The first statement above is quite in harmony with the conditions shown by our tests, viz., the range of softening by recrystallization from the strain-hardened condition is always carried to lower temperatures by prolonging the anneal. In softening, the arsenical metal lags behind the pure copper (temperature-lag of 50° C. for an annealing period of 40 min. in the case of metal containing 0.3 per cent. of arsenic, neglecting the possible effect of silver) and the above result signifies that the stated temperature of 205° C. lies well within the temperature range of softening based upon an annealing period of several weeks in the case of pure copper, while, in the case of the arsenical metal, it lies below, or barely within, the temperature range of softening based upon a similar annealing period.

It is perfectly clear, without undertaking special experiments involving variation of the annealing period, that arsenical coppers, such as the Copper Range product, constitute a commonly available and satisfactory material—superior to pure copper—when it is desired to preserve a rolled temper in the face of considerable variation of temperature up to some 200° C.

The second statement taken from Mr. Johnson's paper, which seems to indicate a difficulty in annealing the arsenical metal, cannot be brought into harmony with the results of our tests. We find that the separate temperature ranges of early softening for the three kinds of material end at a very nearly common (minimum) temperature of complete softening (some 350° C.), from which point onward there are only minor differences in the annealing characteristics of the different kinds of metal. It is inconceivable that any of these products should lose only 6 per cent. of their strength by annealing for a period of hours at red heat. Such an effect must be attributed to the somewhat higher arsenic content (of doubtful application), or to other unknown peculiarities of composition.

As far as the present comparisons go, we are able to say that the proper annealing temperature for electrolytic copper (dull red) will produce equally satisfactory results in the case of Copper Range or Mohawk copper.

Outstanding features of comparison between the three varieties of copper in the annealed condition may be summarized as follows:

[13] F. Johnson: The Influence of Impurities on the Properties of Copper, *Metallurgical and Chemical Engineering*, vol. 8, pp. 570–575 (October, 1910).

Only minor differences in strength throughout the entire annealing range of 400 to 1,000° C. were observed. As a rule, after a given anneal, the difference between extreme values of tensile strength (averages) was less than 1,000 lb. per square inch.

The Copper Range product consistently showed the highest values of elongation and reduction of area and the electrolytic product the lowest, with the Mohawk product occupying a more or less intermediate position.

It is important to observe that, in the temperature region of deterioration, which begins in the neighborhood of 700° with all three varieties, the relative ductilities of the Copper Range and electrolytic products are such that the former will stand an extra 100°, or so, of overheating before it drops to the ductility value of the latter. This means that if the metal is allowed to come to a bright red heat in annealing, a Copper Range load may be expected to emerge with about the same ductility as an electrolytic load annealed at the proper temperature of low redness, while an electrolytic load under the same conditions would presumably fall below this standard.

It will be noticed that Grard's curve of tensile strength practically coincides with our curves after an annealing temperature of about 325° is reached, while Bardwell's curve lies somewhat above ours, as would be expected in the case of wires, but rises in the region of high temperature anneals for some obscure reason. The tests from these different sources show marked discrepancies with respect to the temperature ranges of initial softening. Since the location of this range (*zone de détente* in Grard's phraseology) depends upon the period of anneal and the degree of preliminary cold-working, certain discrepancies would be expected, although hardly of the order shown in these comparisons. As may be seen from the initial strength values, both Grard and Bardwell started their tests with material which had been more severely reduced (by rolling or drawing) than our material. Making allowance for the higher strengths commonly shown by wires, it appears that the materials used by both of these authors had received approximately the same degree of cold-working and, under similar annealing conditions, both should show a lower range of initial softening than our material. Bardwell kept his wires 20 min. in the furnace and Grard kept his strips 30 min. in the furnace, as far as we can determine from his general description of procedure. Our material was reduced 50 per cent. by rolling and we used an annealing period of 40 min., actual time of heating at the annealing temperature.

We do not know how far the presence of a small amount of silver might have effected Bardwell's results, but, on a basis of equivalent purity, the temperature ranges of initial softening found by both authors should not be greatly dissimilar and they should lie somewhat behind the corresponding range found by us in the case of electrolytic copper. As matters stand, the range determined in our experiments occupies an

intermediate position. Not enough definite information is at preser available to permit an harmonious interpretation of these irregularitie: It is particularly to be desired that determinations of the equilibriur positions of the temperature range of initial softening of rolled electrc lytic copper for various reductions be made.

Our curves of elongation similarly occupy an intermediate positio between those of Bardwell and Grard. Owing to the divergent ratic of test length to diameter (or to $\sqrt{\text{sectional area}}$), the actual values c elongation vary within wide limits. A purely fortuitous coincidence be tween Grard's and Bardwell's values is noted at annealing temperature above 400°. This is due to the fact that Grard used a ratio of approxi mately 22 to 1 (length to $\sqrt{\text{sectional area}}$) and Bardwell, a ratio of ap proximately 150 to 1 (length to diameter), which choice of dimension appears to have compensated for differences in the stretch of the tw materials, yielding the same percentage values of elongation.

All of the curves show practically the same temperature range c deterioration (*zone de fléchissement*, in Grard's phraseology), viz., fror 700° upward.

(3) *Tests after Annealing under Reducing Conditions*

Bengough and Hill[14] obtained some strikingly destructive effects b annealing certain arsenical mixtures in reducing gases, which led ther to seek for an explanation in the constitutional relationships betwee copper, arsenic, and oxygen. Further tests by the same authors an also by Johnson[15] and others have established the fact that both arsenica and non-arsenical coppers containing oxygen suffer deterioration b "gassing" when given a reducing anneal above moderate red heat, whil deoxidized copper is not subject to this disadvantage. It is thus reasor ably certain that the effect in question is due to an ordinary reductio of cuprous oxide by hydrogen or other gases which can permeate (*i.e* dissolve in) solid copper, with formation of (bulky) gaseous products thereby causing fissures and cavities to develop.

We have endeavored to ascertain the relative destructive effects whic occur on annealing the three kinds of copper in hand under typical re ducing conditions at several representative temperatures. Accordingly the test strips were annealed at 600°, 800°, and 1,000° C. in an iron tub through which coal gas was passed in sufficient quantity to burn freel at the discharge end, notwithstanding a certain amount of decompositio with formation of soot, in the tube.

On examination of Table 4, in which the results of these tests are t

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[14] G. D. Bengough and B. P. Hill: The Properties and Constitution of Coppe Arsenic Alloys, *Journal of the Institute of Metals*, vol. 3, pp. 34–97 (1910).

[15] F. Johnson: A Method of Improving the Quality of Arsenical Copper, *Journ of the Institute of Metals*, vol. 10, pp. 283–4 (1913).

be found, it will be seen that, of the whole lot of strips tested, only one set, viz., that composed of strips from the Copper Range material annealed at 600°, showed a fair retention of ductility. Here, an elongation of 42 per cent. and a reduction of area of 39.3 per cent. replace the normal values of 56.9 and 50.15, respectively, attained after an oxidizing anneal at the same temperature. The other materials after anneal at this temperature gave unsatisfactory breaks and lower values of ductility, as far as could be ascertained. Destructive effects—brittleness and an almost entire lack of elongation—were obtained throughout in the case of reducing anneals at 800° and 1,000°. As far as may be judged from the small number of tests in which good breaks were secured, the anneals at 1,000° were somewhat less destructive than the anneals at 800°. This was doubtless due to the protective influence of a dense coating of carbonaceous material which was deposited on the strips at the highest annealing temperature.

All fractures showed a lack of luster an and approximation to the brick-red color of fractures from "set" copper.

It seemed of considerable interest to determine the oxygen content of the "gassed" strips in the usual way, viz., by ignition in hydrogen. Accordingly, representative samples were sent to the laboratory of the Michigan Smelting Co. for this purpose. Table 5 embodies the results of analyses from this source.

TABLE 5.—*Oxygen Content of Annealed Copper*

No. of Test Strip (See Table 4)	Kind of Copper	Temperature of Anneal, Degrees Centigrade	Oxygen Content,* Per Cent.
278	Electrolytic	600	0.0456
280	Mohawk	600	0.0441
282	Copper Range	600	0.0402
284	Electrolytic	800	0.0035
286	Mohawk	800	0.0060
288	Copper Range	800	0.0082
290	Electrolytic	1,000	0.0054
293	Mohawk	1,000	0.0010
294	Copper Range	1,000	0.0053

In view of the small percentages of oxygen found in the material annealed at 800° and 1,000°, it is obviously not justifiable to attach great importance to the differences which may be observed in comparing the several results. The analytical data are in general agreement with the testing results. In particular, the figures indicate partial deoxidation of the material at 600° and almost complete deoxidation at 800° and 1,000°.

* The initial oxygen content of the different kinds of copper was as follows: Electrolytic, 0.071 per cent.; Mohawk, 0.052 per cent.; Copper Range, 0.055 per cent.

In general, the tests in this section indicate that the Copper Range material is somewhat less susceptible to the evil effects of a reducing anneal than either of the other varieties of copper. On the other hand, they show that none of these coppers will retain its ductility unimpaired after a 40-min. period of anneal under actively reducing conditions at a temperature even as low as 600°.

It is clearly of great importance to regulate carefully the combustion in all furnaces used for heating copper preparatory to hot-rolling, or for annealing copper after cold-working (unless the metal has previously been deoxidized).

Microstructure

Pieces were detached from the ends of the test strips, representing each separate anneal, for microscopic examination. Photomicrographs are reproduced in the accompanying plates in sets of three (electrolytic, Mohawk, and Copper Range copper, respectively, to correspond with the separate anneals at intervals of 100°, beginning with 200° and ending with 1,000°.

Figs. *a*, *b*, and *c*, of Plate 1, taken from strips of electrolytic, Mohawk, and Copper Range copper, respectively, annealed at 200°, are thoroughly representative of the structure shown by the cold-worked metal and by metal annealed at any temperature short of the softening point. The ordinary characteristics of severe deformation, viz., elongation of the grains in the direction of rolling and wavy lines or bands of deformation at right angles to the direction of rolling, are plainly evident in these photomicrographs.

The curves of tensile strength, elongation, and reduction of area (Figs. 8, 9, and 10, respectively) call for far-reaching recrystallization of the electrolytic copper and hardly more than incipient recrystallization of the other varieties at the next temperature of anneal, 300°. Accordingly, Fig. *d*, Plate 1, shows the presence of numerous minute secondary grains along with unaltered fragments of the primary grains—some of them still marked by lines of deformation, while Figs. *e* and *f* show very little evidence of recrystallization at this magnification (×68).[16]

Complete recrystallization has occurred in all cases at 400° and Plates 2 to 5, inclusive, show the growth of grain which takes place as the temperature of anneal is increased by increments of 100° between 400° and 1,000°.

Baucke[17] has published counting data which indicate that grain

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[16] A satisfactory examination for the early appearance of recrystallization can only be made at moderately high magnifications. For the sake of uniformity, however, we have prepared all photomicrographs at the magnification which has proved most serviceable in the general work.

[17] H. Baucke: On Some Recent Micrographical Investigations of Copper, *International Association for Testing Materials, 6th Congress (New York)*, Section II, vol. 2.

growth on annealing strain-hardened copper (of unknown purity) increases gradually up to 700°, but much more rapidly in the temperature region, 700 to 1,000° C.

The photomicrographs of the present set show conditions quite in harmony with the above generalization which may thus be extended to include the three varieties of copper used in these tests.

We have not undertaken to prepare counting data from these photomicrographs, believing that the principal elements of difference are such as to render qualitative comparisons more valuable than attempted quantitative ones. To take the extreme cases, it is difficult to compare the grain size of the electrolytic samples with that of the Copper Range samples, because of greater variations in the former material. These major variations are due to the presence of finely subdivided oxide particles which are concentrated in certain portions of the copper matrix. In such regions, the mere presence of the oxide particles hinders coalescence between adjoining grains and reduces the final grain size.[18]

The form and distribution of these oxide segregations, as well as the relative conditions presented by the two kinds of copper, are best discussed with reference to their genesis in the cast material.

Fig. *b*, Plate 6, shows at the left a section through primary dendrites of copper in a dark ground mass of eutectic. Planimetric analysis shows an oxygen content of 0.14 per cent.; equivalent to 1.26 per cent. of cuprous oxide. From the regularity of arrangement, it is probable that most of the light areas are parts of the same dendritic growth, *i.e.*, that they are not separate grains, but are sections from the same grain with a eutectic filling in the interstices.

If we increase the quantity of primary copper at the expense of the eutectic, as may be done by lowering the oxide content, the light-grain sections increase in area, changing their contours somewhat; and the eutectic filling becomes attenuated until, with an oxygen content of 0.04 per cent. (rather low for commercial tough-pitch copper), a condition similar to that shown at the right in Fig. *b*, Plate 6, is obtained. The relation between the two portions of this figure is easily seen. The filling in of the interstices with pure copper has obscured the principal features of regularity which could be observed at the earlier stage of dendritic growth, but it is highly probable that most of the light sections seen in the right-hand micrograph are parts of the same grain, *i.e.*, possess the same specific orientation.

The eutectic filling is more massive in some places than in others. Where it is concentrated, an intensive rolling operation will result in a relatively broad band of eutectic, *i.e.*, more or less parallel trains of oxide

[18] Mathewson and Caesar, in a paper lately sent to the *Internationale Zeitschrift für Metallographie*, have discussed the effect of oxide on the size of recrystallized grain in pure copper.

particles in their copper matrix, and, where it is attenuated, narrow bands will result.

In the present experiments, we started with a planed cake, 2 in. thick, and ended with test strips 0.064 in. thick. Hence, the cake was elongated to some 30 times its original length as a result of the various rolling operations, and equiaxial patches of eutectic are ultimately replaced by bands of varying breadth and thickness depending upon their original dimensions.

A broad band of this sort is seen in Fig. *a*, Plate 4. It is very noticeable that the copper grains in this region are smaller than in adjacent regions free from oxide. Other bands of varying width may be seen in other photomicrographs of the present collection.

In the case of the Copper Range material, as cast (Fig. *a*, Plate 6), three important structural observations may be made: (1) there is no eutectic, (2) a cored structure (shadowy markings) appears, showing variation in internal concentration with respect to arsenic, and (3) the individual oxide particles, which are located with some regularity in the ultimate interstices corresponding to final solidification between and within the grains, are considerably larger than the eutectic particles of oxide in the electrolytic material.[19]

The early annealing of this metal removes all traces of the cored structure and no indication of variable internal composition was observed in the product as received from the rolling mill.

Some fundamental questions of constitution in the system, copper, arsenic, oxygen, have not yet been answered. It is clear that, up to some 3 per cent., the arsenic exists as a solid solution, probably of copper arsenide (Cu_3As) in copper[20] and that arsenic does not deoxidize copper, but exerts a coarsening and scattering effect on the oxide particles,[21] as mentioned above. This latter effect may be due to a greatly increased solubility of cuprous oxide in arsenical copper, whereby the whole amount of oxide (present in tough-pitch metal) remains dissolved until almost all of the copper is frozen, and finally separates out in coarse form from this concentrated solution (as a eutectic containing an unusually high percentage of oxide).

Whatever the constitutional reasons for these facts, it is quite evident that the trains of oxide particles, formed when Copper Range metal is rolled, are always made up of coarse units which do not oppose grain

[19] These could not, as a rule, be seen as separate particles under the magnification generally adopted. They are closely sprinkled throughout the dark structure element (eutectic) shown in both parts of Fig. *b*, Plate 6. The conclusions reached above were verified by careful examination of the samples in question at adequate magnifications.

[20] G. D. Bengough and B. P. Hill: The Properties and Constitution of Copper-Arsenic Alloys, *Journal of the Institute of Metals*, vol. 3, pp. 34–97 (1910).

[21] P. Jolibois and P. Thomas: Du Rôle de l'Arsenic dans les Cuivres industriels, *Revue de Métallurgie*, vol. 10, pp. 1264–1270 (1913).

growth as effectively as the highly dispersed form of oxide present in electrolytic copper. Consequently, irregularities of distribution do not, in this case, result in great local variations of grain size, as in the case of electrolytic copper.

The principal conclusions drawn from our structural observations are:

(*a*) From the standpoint that uniformity of recrystallized grain size is a mark of quality in an annealed product, metal rolled from the ordinary Copper Range cake is superior to metal rolled from the ordinary electrolytic cake containing about the same percentage of oxide.

(*b*) With regard to the specific effect of included oxide particles on the properties of the finished sheet, bar, or tube, if a small number of coarse particles are less harmful than a large number of fine particles, the Copper Range product is superior to the electrolytic product. Jolibois and Thomas[22] take this view as a matter of course. We are not prepared to make any definite assertion in this respect. It is not improbable that with one form of oxide distribution the metal may be particularly adaptable to one type of service, while the other form may be preferable when the metal must meet requirements of another type.[23]

A fair sample of tube manufactured from Copper Range cake (by hot-rolling, cupping, and closing in) is illustrated by Fig. *c*, Plate 6.

Typical photomicrographs representing the structure of test strips spoiled by annealing in coal gas are shown in Figs. *d* and *f*, Plate 5. The first of these photos shows a sample of electrolytic copper annealed at 800° and photographed at the usual magnification of 68 diameters. This may be compared with Fig. *a*, Plate 4, representing the same material annealed normally at this temperature.

By careful examination of these photomicrographs, it may be seen that many of the boundaries between grains of the "gassed" metal (Fig. *d*, Plate 5) have a dotted appearance, while, in the normally annealed metal (Fig. *a*, Plate 4), these boundaries are close and sharp (the oxide dots are sprinkled at random without regard to the grain boundaries). Observations of this sort are best made at higher magnifications, as illustrated by Figs. *e* and *f*, Plate 5, taken from Copper Range metal annealed at 800° in air and coal gas, respectively. The first figure shows several trains of unaltered oxide particles (magnification, 450), while the last figure shows an abundance of gas cavities located, for the most part, in the grain boundaries. No unaltered oxide (distinguishable by its color, etc.) could be seen in the samples ruined by drastic annealing in coal gas,

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[22] *Loc. cit.*

[23] Doubtless, in every case, the metal is improved by eliminating oxide. Arsenical copper can be deoxided with phosphorous, the commonest of deoxidizing agents, without excessive loss of arsenic. It is worthy of note that the producers of Copper Range copper profess to be able to bring the metal to pitch with a lower content of oxide than is commonly attained in the case of non-arsenical copper.

i.e., in those annealed at 800° and 1,000°, while, in the samples annealed at 600° some unaltered oxide could be seen, along with the gas cavities resulting from reduction of the remainder.

Those who have seen Campbell's[24] recent micrograph of over-poled copper may recall its close resemblance to our last micrograph (Fig. *f*, Plate 5) of copper ruined by anneal in coal gas. Deterioration, in both cases, is due to the development of gas within the grains and along the grain boundaries.

Aside from the comparative observations already described, we have made no special study of the effects of annealing under reducing, or other exceptional conditions. It is beyond the scope of this paper to discuss the results obtained by other authors in this particular field.

Our observations indicate that the oxide contained in any of these brands of copper is decomposed by coal gas, even at temperatures of low redness. Presumably, such components of coal gas as can travel freely in copper at red heat (certainly hydrogen, probably carbon monoxide, and perhaps certain hydrocarbons) effect an ordinary reduction of the oxide, with formation of bulky gaseous products which seem to accumulate in the grain boundaries,[25] but also occur abundantly within the grain substance.

The character of the fractures indicates that the tensile break occurs very largely at the grain boundaries through the gas cavities, whose lining seems to be devoid of luster.

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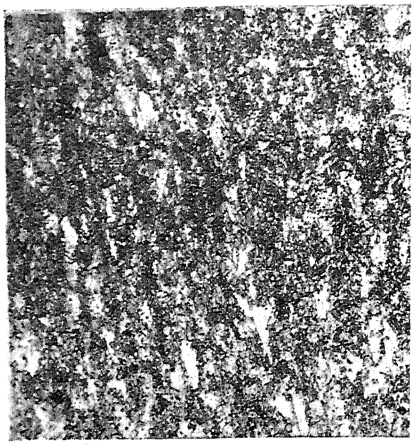
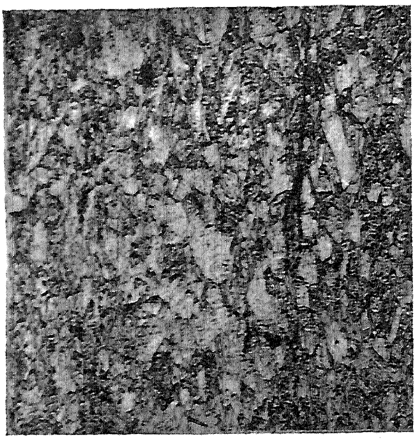
[24] William Campbell, International Engineering Congress, San Francisco, 1915.

[25] G. D. Bengough and D. Hanson's tensile tests (*Journal Institute of Metals*, vol. 12, pp. 56–88, 1914) have shown that these boundaries are favored regions of weakness at high temperatures (the grains pull apart at the boundaries), which indicates that gas evolved from within would find its path of least resistance along the boundaries.

a.—Electrolytic Copper, 200° C. × 68.

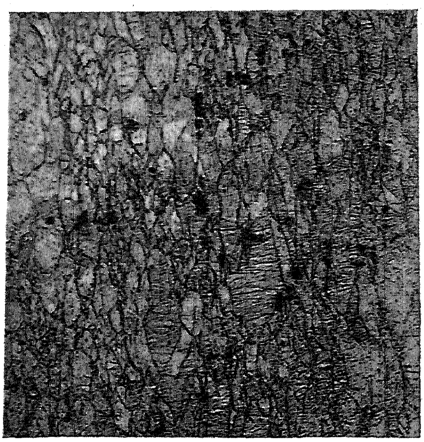


d.—Electrolytic Copper, 300° C. × 68.

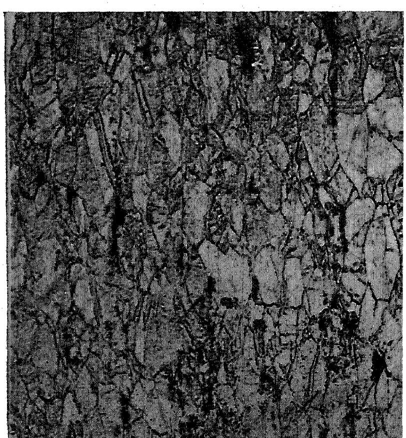


b.—Mohawk Copper, 200° C. × 68.

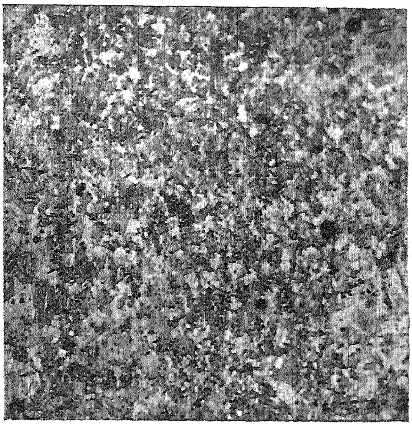
e.—Mohawk Copper, 300° C. × 68.



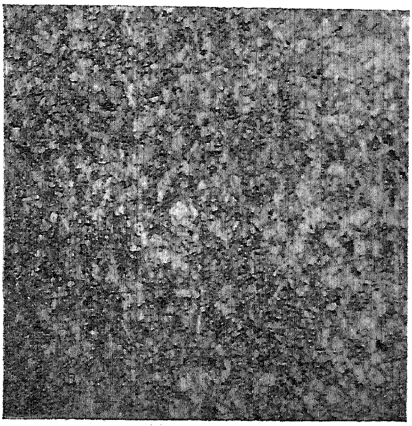
c.—Copper Range Copper, 200° C. × 68.



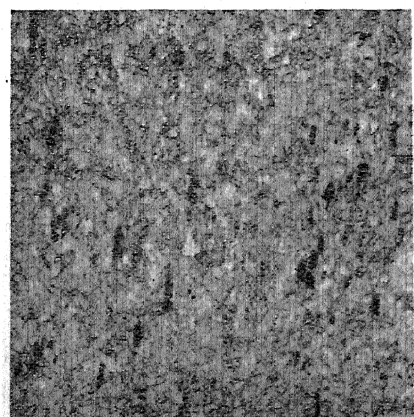
f.—Copper Range Copper, 200° C. × 68.



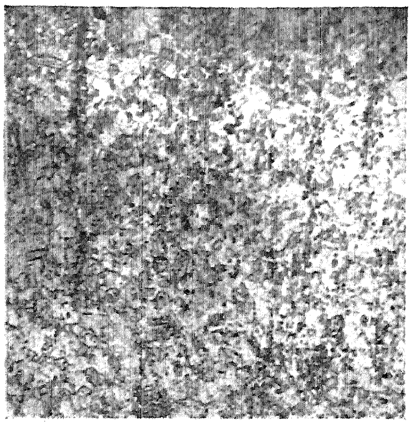
a.—Electrolytic Copper, 400° C. × 68.



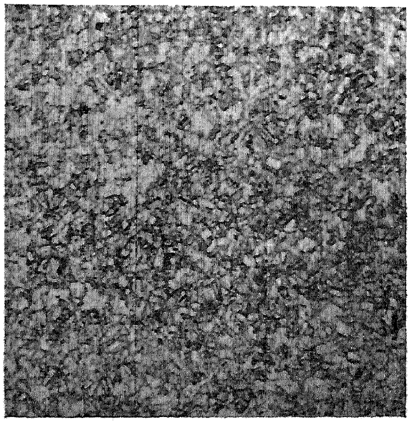
b.—Mohawk Copper, 400° C. × 68.



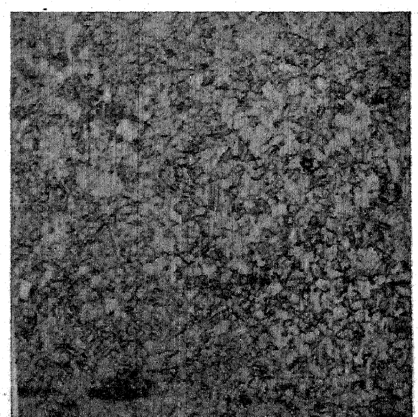
c.—Copper Range Copper, 400° C. × 68.



d.—Electrolytic Copper, 500° C. × 68.

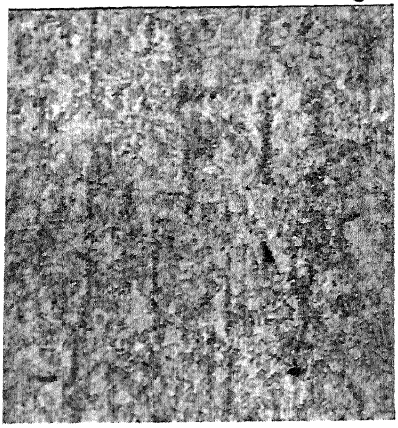


e.—Mohawk Copper, 500° C. × 68.

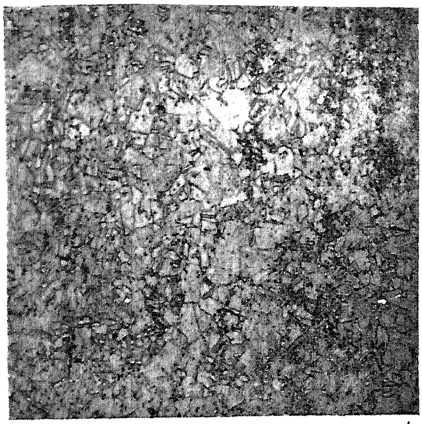


f.—Copper Range Copper, 500° C. × 68.

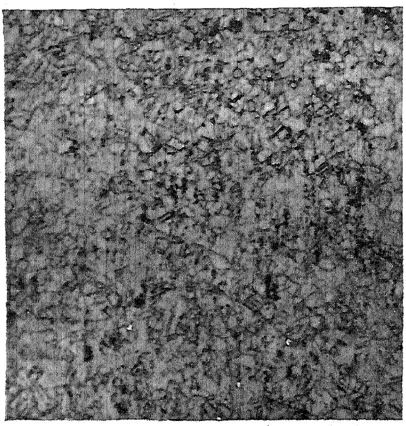
PLATE 2.—DIFFERENT VARIETIES OF COPPER ANNEALED AT 400 AND 500° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.



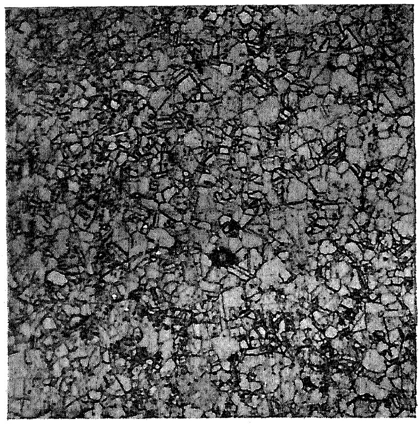
a.—Electrolytic Copper, 600° C. × 68.



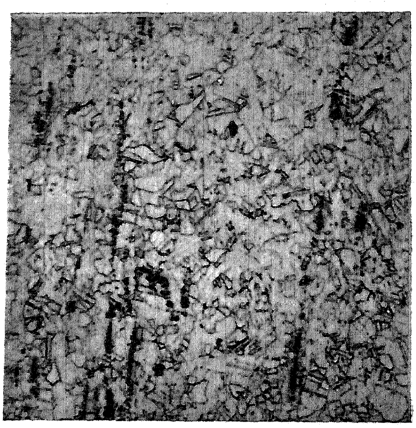
d.—Electrolytic Copper, 700° C. × 68.



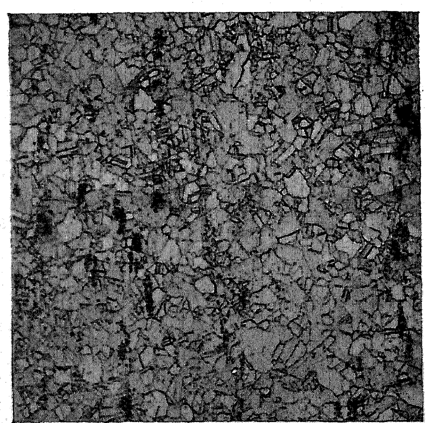
b.—Mohawk Copper, 600° C. × 68.



e.—Mohawk Copper, 700° C. × 68.



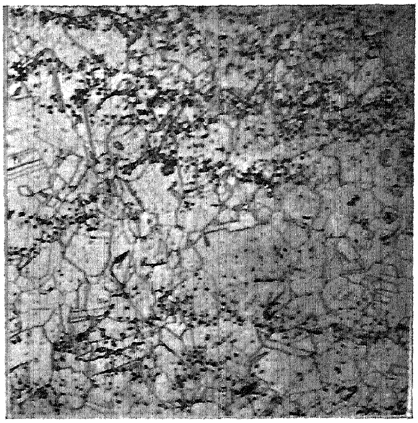
c.—Copper Range Copper, 600° C. × 68.



f.—Copper Range Copper, 700° C. × 68.

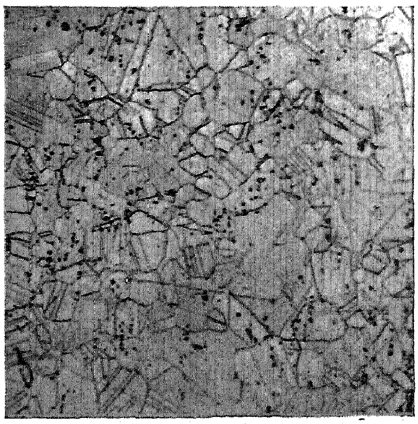
PLATE 3.—DIFFERENT VARIETIES OF COPPER ANNEALED AT 600 AND 700° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.

a.—Electrolytic Copper, 800° C. × 68.

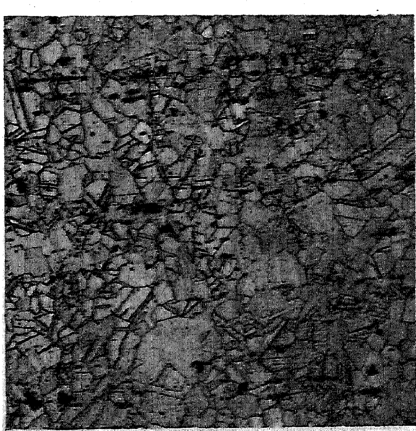


d.—Electrolytic Copper, 900° C. × 68.

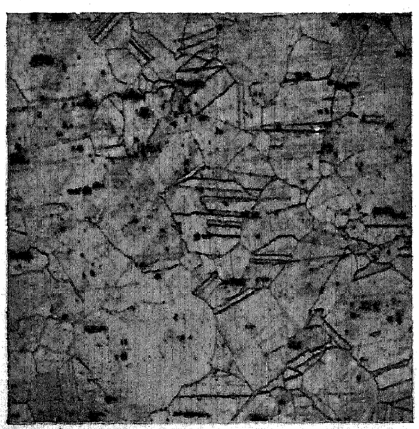
b.—Mohawk Copper, 800° C. × 68.



e.—Mohawk Copper, 900° C. × 68.



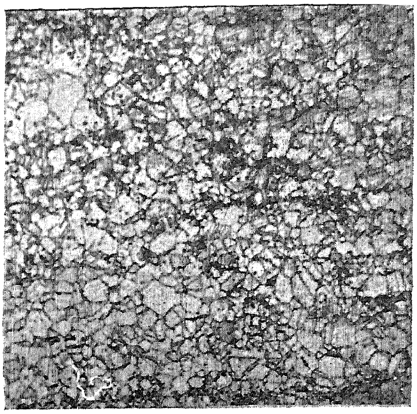
c.—Copper Range Copper, 800° C. × 68.



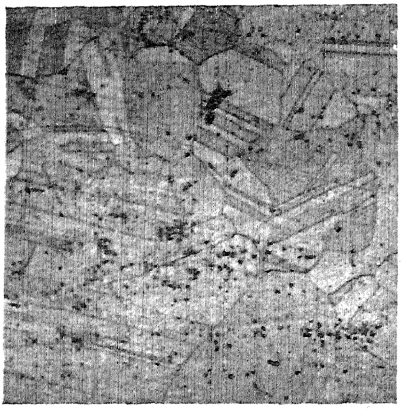
f.—Copper Range Copper, 900° C. × 68.

PLATE 4.—DIFFERENT VARIETIES OF COPPER ANNEALED AT 800 AND 900° C. SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE.

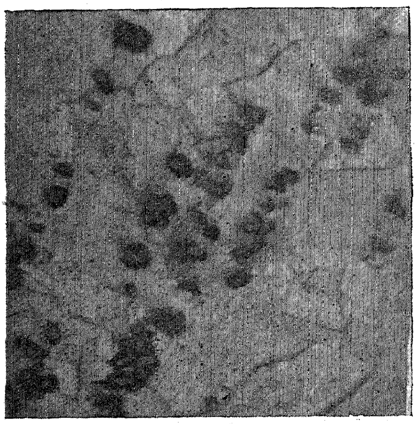
a.—Electrolytic Copper, 1000° C. ×68.



d.—Electrolytic Copper, Annealed at 800° C. in Coal Gas. × 68.

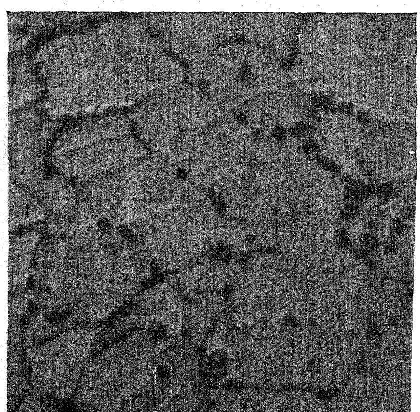


b.—Mohawk Copper, 1000° C. × 68.



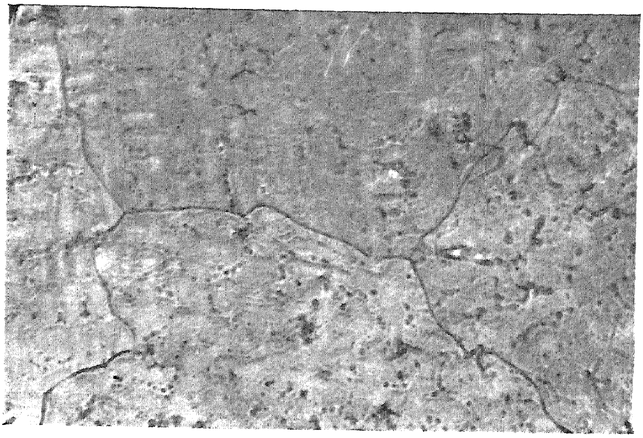
e.—Copper Range Copper, Annealed at 800° C. in Air. × 450.

c.—Copper Range Copper, 1000° C. × 68.

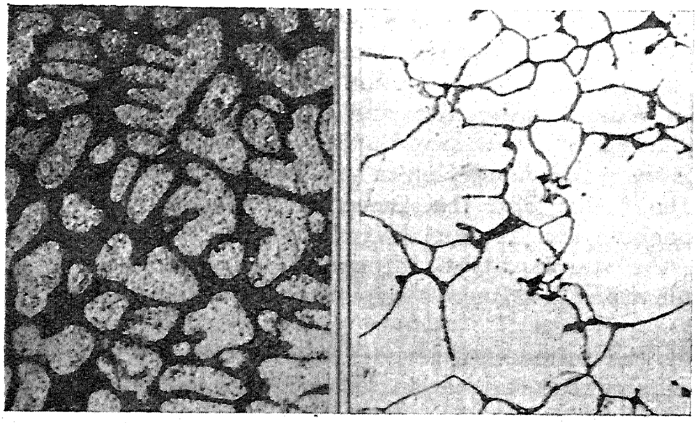


f.—Copper Range Copper, Annealed at 800° C. i Coal Gas. × 450.

PLATE 5.—DIFFERENT VARIETIES OF COPPER ANNEALED AT 1000° C IN AIR, AND AT

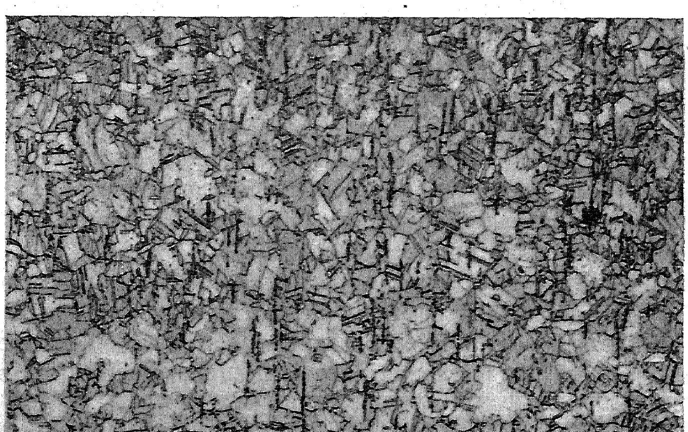


a.—Copper Range Copper as Cast. 0.055 Per Cent. Oxygen.



b.—Electrolytic Copper as Cast.

0.14 Per Cent. Oxygen. 0.04 Per Cent. Oxygen.



c.—Annealed Tube from Copper Range Copper.

PLATE 6.—SAMPLES ETCHED WITH AMMONIA AND HYDROGEN PEROXIDE. × 69.

Flotation Concentration at Anaconda, Mont.

BY FREDERICK LAIST,* B. S., AND ALBERT E. WIGGIN,† B. S., ANACONDA, MONT.

(Arizona Meeting, September, 1916)

I. EXPERIMENTAL FLOTATION CONCENTRATION

INTRODUCTION

EARLY in 1914 it was decided to test, on a fairly large scale, the treatment by flotation of Anaconda slime and mill tailing. For this purpose a standard-type Minerals Separation machine was installed at the Washoe Reduction Works during May and June, 1914. This was followed by the installation of a full-size Callow pneumatic machine plant. Experiments were also made, on a smaller scale, with the Froment, the Towne, the Fields, and the Anaconda flotation machines. The last-named machine was developed at this plant. In addition to the tests made in the standard-type Minerals Separation machine some tests were made using a Minerals Separation machine of the sub-aeration type.

During the series of experiments a large variety of oils was tested. Experiments were also conducted using both round-table feed and tailing to determine whether it would be better to displace the round tables by flotation for the treatment of the slime, or to supplement the round tables by flotation of the round-table tailing.

A series of tests was also made on the treatment of the mill tailing by grinding followed by flotation to determine the relative merits of flotation and leaching for the treatment of this product. In addition, flotation tests were made on mixtures of mill tailing and slime.

The round-table feed referred to above is the total slime from the mill. It contains about 35 per cent. colloidal solids and approximately 90 to 95 per cent. of the total solids will pass through 200 mesh (0.067 mm.). It assays from 2.3 to 2.6 per cent. Cu.

The mill tailing referred to above is the total discard from the mill, exclusive of the slime. It is all finer than 2 mm. and about 90 to 95 per cent. will remain on 0.25 mm. It assays about 0.60 per cent. Cu.

A brief summary of the experimental flotation results follows:

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† Superintendent of Concentration, Anaconda Copper Mining Co.

Preliminary Tests

A series of tests was first carried out to determine roughly the best conditions for flotation, using the standard Minerals Separation machine and treating round-table feed: The following reagents were tested either alone, or in combinations: Turpentine, crude oil, cresylic acid, stove oil, tar oil, Carolina oil of tar, argols, sludge acid, fuel oil, wood creosote, and sulphuric acid. In some of these tests sulphuric acid was used and in others it was omitted. Also, the effect of the temperature of the pulp upon the flotation results was tested by heating to various temperatures.

As these tests were merely preliminary, no record was kept of the amount of reagents used. It was conclusively proved, however, that the best combination of reagents was sludge acid, wood creosote, stove oil, and sulphuric acid. Fortunately, of all the reagents tested, these happened to be the cheapest. It was also proved that the addition of sulphuric acid to the pulp was of decided advantage in the treatment of the slime. In two successive tests in which sludge acid, wood creosote, and stove oil were used, the tailing assayed 1.25 per cent. Cu when no acid was used and 0.3 per cent. Cu when acid was used. Since these tests were made we have omitted the use of stove oil.

A. Tests with Standard Minerals Separation Machine

This machine, with the accessory apparatus, was installed in a separate building, south of the round-table plant. It had 16 agitator compartments, each 2 ft. square, and 14 spitzkästen, and was of the standard Minerals Separation design. This machine is known by us as M. S. Machine No. 1. The agitators were of the standard Minerals Separation type, the impellers being 18 in. in diameter and the agitators making 265 r.p.m. This gave the impellers a peripheral speed of 1,245 ft. per minute. The machine required 45 to 55 hp., including motor and belt transmission loss, when operating under a full load of slime pulp.

We wish at this point to express our appreciation of the able manner in which the experimental work on the Minerals Separation machine was carried out by George A. Chapman, and staff, of the Minerals Separation Co.

The first products to be tested were the round-table feed and tailing.

1. *Treatment of Round-Table Feed and Tailing*

(a) Round-Table Feed. Normal Tonnage (60 *Tons*). *Pulp Heated to* 70° *to* 90° *F.*—The reagents used in Periods 2 and 11 were sludge acid kerosene (M. S. 37), crude wood creosote (M. S. 33), stove oil (M. S. 8), and sulphuric acid. The reagents in Period 20 were the same, except

that the wood creosote was obtained from the Pensacola Tar & Turpentine Co., instead of from the Cleveland-Cliffs Co. The creosote from the Cleveland-Cliffs Co., known as M. S. 33, is a derivative from hard pine, while that from the Pensacola Co. is derived from a soft pine. The sludge acid kerosene is a byproduct from the distillation and purification of kerosene, and that known as M. S. 37 is derived from California crude oil by the Union Oil Co. The stove oil, known as M. S. 8, is from the Standard Oil Co. of California, and seems to be largely kerosene.

Period 2, in which the pulp was heated to an average temperature of 89° F., gave a tailing running 0.23 per cent. Cu, with a fair grade of concentrate.

Period 11, in which more creosote was used, all other conditions remaining as in Period 2, gave a tailing assaying 0.27 per cent. Cu, with a slightly better grade of concentrate. It should be noted, however, that it is not characteristic of creosote to give a clean concentrate, it being rather a good "tailing" oil and the slight increase in the grade of concentrate should not be attributed to the increase in the amount of creosote used, but rather to a difference in operation of the machine.

Period 20, in which Pensacola creosote was used, shows an average tailing of 0.33 per cent. Cu, and a very clean concentrate. It should be noted that considerably less creosote was used than in Period 11. The decided increase in the grade of concentrate is not due to the substitution of Pensacola creosote for Cleveland-Cliffs creosote, but rather to differences in operation.

(*b*) *Round-Table Feed. Pulp not Heated.*—During Periods 3 and 10, the pulp was not heated, but treated at a temperature of 50° F. The tests were made during the summer months; it should be noted that during the winter months the temperature of the pulp would be about 40° F., with a minimum of about 35° F. During Period 3, the tailing assayed 0.32 per cent. Cu, with a low-grade concentrate, and during Period 10 the tailing averaged 0.42 per cent. Cu, with a higher grade concentrate.

(*c*) *Round-Table Feed. Pulp Heated to* 125° *F.*—Heating the pulp to a temperature of 125° F. (Period 4) gave a tailing of 0.21 per cent. Cu with a fair grade of concentrate. As Period 2, in which the pulp was heated to 90° F., gave a 0.23 per cent. Cu tailing, it is evident that heating beyond 90° F. does not pay. In fact, a careful study of the tabulated results would indicate that it does not pay to heat above about 75° F. The cost of heating the slime pulp for flotation is about 1c. per ton of dry slime for every 2½° F.

(*d*) *Round-Table Tailing. Pulp not Heated.*—The treatment of the round-table tailing, after dewatering to a density of 11.5 per cent. solids (Period 5), gave a tailing of 0.25 per cent. Cu with a low-grade concentrate. It was thought that possibly the net recovery would be sufficiently more to justify the retention of the round tables if the tailing

from the round table instead of the feed to the round table were treated by flotation.

(*e*) *Round-Table Tailing. Pulp Heated to* 106° *F.*—It is evident from the result of this test (Period 6) that the heating in the case of the round-table tailing was of no benefit.

(*f*) *Round-Table Feed. High Density of Pulp. Pulp Treated to* 96° *F.* —The density of the round-table feed was increased to 22.5 per cent. solids by a second dewatering (Period 7). As had been expected, the increase in density considerably decreased the oil consumption without decreasing the recovery of copper, and of course would also decrease the expense of heating the pulp.

(*g*) *Round-Table Feed. High Density of Pulp. High Tonnage* (105 *Tons*). *Pulp Heated to* 96° *F.*—It was thought that by increasing the density of the pulp it might be possible to treat the same volume of pulp per machine unit as with the lower density, and in that way increase the tonnage of solids treated. The tests under Period 8, however, show that any considerable increase in the amount of feed to the M. S. No. 1 machine over 60 tons results in a decided increase in the tailing loss.

(*h*) *Round-Table Feed. High Density of Pulp. High Tonnage. Pulp not Heated.*—The value of heating the pulp in the case of high-density feed pulp is brought out by this test (Period 9), where the tailing jumped to 1.28 per cent. Cu with the absence of heat.

(*i*) *Round-Table Tailing. High Pulp Density. Pulp Heated to* 70° *F.*—This test (Period 12) gave a tailing of 0.45 per cent. Cu, and a low-grade concentrate.

(*j*) *Round-Table Tailing. Medium Pulp Density. Pulp Heated to* 80° *F.*—This test (Period 13A) gave a good tailing assaying but 0.20 per cent. Cu. This would seem to indicate that too high a pulp density in the treatment of the round-table tailing is a decided disadvantage.

(*k*) *Round-Table Tailing. Medium Pulp Density. Pulp Heated to* 80° *F.*—In this test (Period 13B) no acid was used, and as a result the tailing went up to 0.96 per cent. Cu. Note also the low silver content of the concentrate.

(*l*) *Round-Table Tailing. Low Pulp Density. Pulp Heated to* 70° *to* 80° *F.*—In Periods 14A, B, C, and D, the round-table tailing pulp was not dewatered at all, but treated directly at a density of about 6 per cent. solids. The tailing was good, ranging from 0.16 per cent. Cu to 0.22 per cent. Cu, for the four tests. During Period 4B, crude turpentine, M. S. 14, from the Georgia Pine Products Co., was used in place of Cleveland-Cliffs creosote; during Period 14C, crude wood creosote, No. 400, from the Pensacola Tar & Turpentine Co., was used, and during Period 14D, crude wood creosote from the Crichton Pine Products Co., Ala., was used. All of these oils gave satisfactory results. The increased amounts of these oils used during these tests were un-

doubtedly due to the low density of the pulp, and to the low tonnage treated.

(m) *Round-Table Feed. Low Pulp Density. Low Tonnage* (40 to 50 *Tons*). *Pulp Heated to from* 70° *to* 80° *F.*—It was thought that the low tailing produced during Periods 14A, B, C, and D was due to the low tonnage treated, and possibly to the low pulp density, as well as to the fact that round-table tailing was being treated. Accordingly, the tests under Periods 15A, B, C, D and E were made and in these a low tonnage of round-table feed was treated with a low pulp density. These tests gave fully as good results from the recovery standpoint in a considerably better grade of concentrate as those made on the round-table tailing under similar conditions.

During Period 15A, wood creosote from the Crichton Pine Products Co. was used. During Period 15D, no creosote was used and the tailing assay increased from about 0.20 per cent. Cu to 0.32 per cent. Cu. During Period 15E, some creosote from the Rocker Timber Treating Plant, near Butte, was used, but gave a high tailing—0.42 per cent. Cu. During Period 15B, tar creosote from the Butte Gas Co. was used, with apparently good results, although the test was too short to be taken as conclusive proof of the value of this creosote as a flotation oil.

(n) *Round-Table Feed. Low Tonnage* (50 *Tons*). *Pulp Heated to* 70° *F.*—The tests (Period 16A) gave a tailing running 0.21 per cent. Cu, and a fair grade of concentrate.

(o) *Round-Table Feed. Pulp Heated to* 70° *F. High Speed of Agitators.*—It was thought that by increasing the speed of the agitators the amount of feed treated might be increased without decreasing the recovery. The first nine agitators were used and their speed was increased from 265 r.p.m. to 363 r.p.m. The test (Period 16B) was of short duration, as the machine would have racked itself to pieces if operated very long at this high speed. While the test is too short to be conclusive, the indications are that the increased speed was a detriment rather than an advantage. It is thought that there is a critical speed at which the agitators should be run for the most efficient operation. If we go above this speed the power increases much faster than the capacity, and if we go below it we do not get a proper mixing of oil and acid with the pulp. With the agitators running at this speed, the nine cells required 56 hp. including motor and belt transmission loss.

(p) *Round-Table Tailing. Low Tonnage* (40 *Tons*). *Pulp Heated to* 70° *F. Low Pulp Density.*—These tests (Periods 17 and 18) show average tailings of 0.22 per cent. Cu and 0.26 per cent. Cu respectively. The concentrate is rather low grade.

(q) *Round-Table Feed. High Tonnage* (90 *Tons*). *Pulp Heated to* 70° *F.*—The tests under Period 19 gave a high tailing, 0.43 per cent. Cu, and a good grade of concentrate. Some wood creosote from the

Pensacola Tar & Turpentine Co. was used together with that from the Cleveland-Cliffs company. The high tailing is due, however, entirely to overloading the machine.

Under Period 23 the tonnage treated was just as high, the grade of concentrate only a little lower, and yet the tailing is good, 0.25 per cent. Cu. This apparent discrepancy between these two tests is explained, it is thought, by the fact that the slime produced in the mill was considerably more granular during the tests under Period 23 than during those under Period 19.

(*r*) *Round-Table Feed. Low Tonnage* (40 *Tons*). *Pulp Heated to* 70° *F.*—In this test (Period 21), the tailing averaged 0.25 per cent. Cu and the concentrate was medium grade. No Cleveland-Cliffs creosote was used, all the creosote used being from the Pensacola company. The tailing is practically no better with this low tonnage (40 tons) than under Period 23 with the high tonnage (90 tons).

(*s*) *Round-Table Feed. High Tonnage* (90 *Tons*). *Air Used in Last Spitzkasten.*—It was thought that the introduction of air in the bottom of the last spitzkasten might increase the capacity of the machine and at the same time maintain the same mineral recovery. At first it seemed as though the introduction of the air (Period 22) had resulted in a considerable increase in the capacity of the machine, but when the tests under Period 23 were made, in which the air was omitted, it was found that it was not the introduction of the air which had enabled us to increase the tonnage without decreasing the recovery.

(*t*) *Conclusions.*—1. The economic capacity of the M. S. No. 1 machine when treating slime as produced from the mill at present (May 1, 1915) seems to be from 80 to 90 tons.

2. The best combination of reagents for the treatment of slime seems to be sulphuric acid, kerosene sludge acid, wood creosote and stove oil. There is some question as to the real value of the stove oil. Its principal function seems to be to make a more compact froth.

3. It would not be economical to retain the round tables as the recovery by treating the slime directly by flotation is just as high as by retaining the round tables and treating the round-table tailing by flotation. The grade of concentrate would probably be the same in either case, but any difference would be in favor of treating the round-table feed directly by flotation. The heating of the round-table tailing pulp, on account of its low density, would increase the cost of the flotation.

4. In treating the round-table feed directly by flotation, the resulting tailing should assay 0.30 per cent. Cu, or less, with a concentrate carrying not over 40 per cent. insoluble. Possibly the concentrate can be made much cleaner with no sacrifice in the recovery.

5. It is thought that the best circuit density for the slime pulp for flotation treatment is about 12 per cent. solids.

6. It is thought that about 70° F. will be found to be the most economical temperature at which to keep the pulp.

7. Acid seems to be absolutely essential to the successful treatment by flotation of our slime.

8. The addition of air in the last spitzkasten is of no advantage.

9. Any considerable increase in speed of the agitators above a peripheral speed of about 1,300 ft. per minute seems to be disadvantageous.

2. *Treatment of Mill Tailing after Grinding through* 60 *Mesh*

These tests were made in the M. S. No. 1 machine. Mill tailing from Sections 7 and 8 of the concentrator were elevated and then dewatered. The dewatered tailing was then crushed through 60 mesh (0.25 mm.), in either a Hardinge mill 10 by 4 ft., or a tube mill 8 by 12 ft. The grinding mills were operated in closed circuit with a Dorr classifier, the overflow of the classifier being the final product of the system and going to the flotation plant for treatment.

Following is a screen sizing test of the average final product produced during the test on the Hardinge mill, and is typical of the flotation feed:

Screen Sizing Test on Dorr Classifier Overflow, or Feed to Flotation Machine

Screen Size		Cumulative Per Cent., Solids
Square Mesh	Aperture, Mm. Square	
+16	1.180	0.3
+24	0.730	1.3
+40	0.430	3.0
+60	0.260	5.8
+80	0.210	12.8
+110	0.130	38.5
+130	0.110	42.3
+160	0.085	54.8
+200	0.076	59.3
+240	0.063	62.8
−240	0.063	37.2

(*a*) *Preliminary Tests.*—These tests (Period 24) were started immediately after putting the Hardinge mill in operation. At first no sulphuric acid was added, and the pulp was not heated. It would be well to note here that the sludge acid kerosene contains from 50 to 60 per cent. sulphuric acid, so that when this oil is used we could not have a non-

acid pulp. We found, however, that the use of acid in addition to that contained in the sludge was of advantage. Some very low tailings were produced during these preliminary tests, but the concentrate was very low grade. It seemed to be of decided advantage to add the oil ahead of the grinding mill, the latter apparently making an ideal agitator.

However, as soon as we began to heat the pulp ahead of the mill, we found that the acid in the oil and the small amount of copper sulphate formed began to corrode the iron work of the mill. This would not be a serious matter if we used a pebble, or silex-block lining, with pebbles for grinding, but we early abandoned the idea of using silex, or pebble linings, and there was a good chance that as a grinding medium we would find iron balls superior to pebbles. This, of course, precluded the use of any acid, or acid oil, ahead of the grinding mill, especially if the pulp was heated.

It is barely possible that we might find the action of the sludge acid kerosene in a cold circuit so slight that it could be neglected, even though we used steel lining and steel balls. It would be a decided advantage to add the oil ahead of the grinder.

(*b*) *Low Tonnage* (125 *Tons*). *Pulp Heated to* 70° *F. Oil and Acid added in M. S. Machine. Acid Circuit.*—The tailing produced during this period (No. 25) averaged 0.07 per cent. Cu. The concentrate was too siliceous, averaging 40.1 per cent. insoluble. The only reagents used were sulphuric acid and sludge acid kerosene. It should be noted that the product treated during this period contained only 3.1 per cent. on 80 mesh (0.20 mm.).

(*c*) *High Tonnage* (187 *Tons*). *Same Conditions as under* (*b*), *except that Flotation Machine was Divided into Two Parts, Making a Primary and a Secondary Machine, the Secondary Machine being used to Clean the Concentrate from the Primary Machine.*—The flow sheet was as follows: The feed pulp was put into the sixth agitator cell, thus making the last 11 cells act as a primary machine. The concentrate from the first six boxes of the primary machine was returned to the first three boxes of the original machine through the two pre-agitators. The overflow from the remaining five boxes of the primary machine was returned to the original feed. The last cell made the usual tailing. The secondary machine made a final concentrate and a middling which was returned to the circuit. The chief reagents used were sludge acid kerosene and sulphuric acid. Small amounts of creosote were used during part of the test, but it seemed to be of no particular advantage. During two days, sludge acid kerosene from the Standard Oil Co. of California was used with good results.

The tailing during this period (No. 26) averaged 0.11 per cent. Cu, and the concentrate averaged 26.9 per cent. insoluble. It is thought that this is about typical of the results we may expect from a commercial

installation. It should be noted that 14.4 per cent. of the flotation feed remained on an 80-mesh screen (0.2 mm.). Finer grinding would undoubtedly result in a lower tailing assay, but we must, of course, set against the increased copper recovery the cost of the finer grinding.

It would be well to call attention to the fact that all averages shown in these flotation tabulations are arithmetical and not geometrical. It was found that the arithmetical average practically checked the geometrical in the test cases, and as the geometrical averages would have involved a great deal of extra work it was considered not worth while to average the results geometrically. For example, Period 26 was averaged geometrically and gave a tailing assay of 0.109 per cent. Cu against the arithmetical average of 0.11 per cent. Cu. The concentrate averaged 7.777 per cent. Cu geometrically, and 7.82 per cent. Cu arithmetically. As the daily variations in tonnage and assays are considerable during this period, we would expect to find the arithmetical and geometrical averages diverging more than during the other periods in which the variations are not so great.

(*d*) *Low Tonnage.* (124 *Tons*). *Same Conditions as under* (*c*), *except Turpentine* (*M. S.* 14) *added ahead of Grinding Mill, and no Sludge Acid Kerosene used. Pulp Heated to about* 70° *F.*—In this test (Period 27) it was endeavored to take advantage of the grinding mill as an agitator and for this reason a neutral reagent, turpentine, was used. The tailing is high, 0.12 per cent. Cu, considering the comparatively low tonnage of feed and the fact that only 5.2 per cent. of the feed was coarser than 80 mesh. The same flow sheet was used as during Period 26.

(*e*) *Low Tonnage* (106 *Tons*). *Oil added ahead of Grinding Mill. Non-Acid Circuit. Pulp Heated to about* 75° *F. Various mixtures of crude turpentine, creosote and pine oil used during these tests. The mixtures contained from* 60 *to* 70 *per cent. crude turpentine;* 21 *to* 32.5 *per cent. creosote;* 2.5 *to* 15 *per cent. pine oil.*—The tailing produced during this period was fairly good, averaging 0.09 per cent. Cu, but the grade of concentrate was not good, averaging 41.5 per cent. insoluble.

(*f*) *Conclusions.*—1. Although not definitely demonstrated, it is thought that the economical capacity of the M. S. No. 1 machine when treating sand tailing crushed through 60 mesh is about 175 to 200 tons per 24 hr.

2. The best combination of reagents seems to be sludge acid kerosene and sulphuric acid. However, a mixture of creosote, turpentine, and pine oil, in a non-acid circuit gave good results also. The non-acid circuit, however, seems to require more delicate adjustment and more careful attendance than the acid circuit.

3. The grinding mill makes an ideal agitator, and it is of decided advantage to add the oil ahead of the grinders.

4. The treatment of the mill sand tailing ground through 60 mesh

should result in a tailing assaying not over 0.10 per cent. Cu and a concentrate carrying not over 30 per cent. insoluble.

5. It is thought that the best density of pulp is from 25 to 30 per cent. solids.

6. Heating of the pulp to about 70° F. seems to be of advantage, although there is a possibility that this heating may be dispensed with during the summer months without any injurious results.

7. Acid seems to be beneficial but it is not of as much importance as in the treatment of the slime.

3. *Treatment of Mixture of Round-Table Feed and Mill Tailing after Grinding through* 60 *Mesh*

These tests were made in the M. S. No. 1 machine. It was thought that it might be of advantage to mix the slime and the reground mill tailing for flotation treatment. The acid sludge kerosene, turpentine and the sulphuric acid used were added in the flotation machine. In some instances, various mixtures of coal tar (70 to 80 per cent.), creosote (17.5 to 22.5 per cent.), and pine oil (2.5 to 7.5 per cent) were used with the sludge acid. These were added ahead of the grinding mill.

The average proportion of sand tailing to slime in the mixture treated was 75.7 to 20.1, or 3.8 to 1. In practice the proportion of production of tailing to slime is about 3 to 1; thus our mixture was somewhat deficient in slime. The concentrate produced, 34.1 per cent. insoluble, is of a good grade, but the tailing is high, 0.20 per cent. Cu. Theoretically, the tailing should have assayed about 0.15 per cent. Cu, assuming a 0.10 per cent. Cu tailing from the sand tailing and 0.30 per cent. Cu tailing from the slime.

Although this test was not conclusive, it was decided, from observation, that it is better to treat the slime and the sand tailing separately. Of course, the slime which is made in the grinding of the sand tailing is included in the sand tailing for treatment. This slime produced in grinding the tailing is much lower grade and more siliceous than the original mill slime.

B. Tests with Minerals Separation Machine of Sub-Aeration Type

1. *Treatment of Round-Table Feed and Tailing*

This machine was sent here by the Minerals Separation Co., and was set up and tested at its request.

The principal difference between this and the No. 1 machine is that air is introduced at the bottom of the agitator compartment. The agitator was of the Howard type, practically a "balanced" pump im-

peller run backward, and each shaft had two impellers, one placed above the other and separated by a stationary grid. There was also another grid above the top impeller. There were four agitator compartments and four spitzkästen. The agitators made 185 r.p.m. At first it was planned to suck air from the atmosphere by the action of the impellers. This did not work, however, and it was found necessary to force air into the cells under a few pounds pressure. The feed was introduced through the bottom of the first agitator cell. The laundering was arranged so that as many spitzkästen as desired could be sent to finished concentrate, the remainder going to middling and being returned to the machine. This machine required 47 hp. under a full load of slime, including motor- and belt-transmission loss.

Two tests were made, one treating round-table feed, and one treating round-table tailing. The results of these tests are shown in Periods 30 and 31 in the tables. It may be possible to develop an efficient machine of this type, but the particular machine sent here for testing was certainly not satisfactory.

A later test was made in which the spitzkästen were dispensed with and the froth was taken directly off the top of the agitator cells. Under these conditions, the four agitators seemed to create too much disturbance for the proper removal of the froth, and only two agitators, the first and third, were operated. These modifications did not improve the work of the machine. The results of this test are shown below. This modified machine was called No. 2 A.

C. Tests with Callow Pneumatic Machine

Tests made by Mr. Callow at his laboratory in Salt Lake on samples of our mill tailing ground through 40, 60, and 80 mesh, and of our slime, had given such promising results that it was decided to try out the Callow machine on a commercial scale. Accordingly, there was shipped here during September, 1914, five standard Callow cells, 2 by 8 ft., a Pachuca agitating tank and accessory apparatus, consisting of blower and sand pumps. This equipment was installed in the old 80-ton experimental leaching plant and was ready for operation the latter part of October.

In addition to the Pachuca agitator recommended by Mr. Callow, we built a set of two mechanical agitators. These agitators consisted of a tank about 10 ft. long by 2½ ft. wide and 2½ ft. deep, in which revolved a horizontal shaft carrying a set of paddles. These agitators were belt driven from one motor and required a total of 25 to 30 hp., including motor, belt, and countershaft power loss. The agitators seemed to work well and had a combined capacity of about 60 tons of slime per 24 hr.

Power readings made on the blower which furnished the air for the five Callow cells gave the following results:

Air Pressure, Pounds	Input to Motor, Hp.	Shafting, Hp	Net to Blower,* Hp.
4	25.8	8.8	17.0
5	35.4	8.8	26 6
6	47.4	8 8	38.6

* Includes motor loss.

1. *Treatment of Round-Table Feed and Tailing*

(*a*) *Round-Table Feed. No Acid.*—During the first four days air agitation was employed, using the Pachuca tank. With a feed of 60 to 80 tons, the results were very poor. The air agitator did not have sufficient capacity. During the last five days of this period (No. 32) mechanical agitation was used and gave much better results.

(*b*) *Round-Table Feed. Air Agitation.*—This test (Period 33), which was of 16 days' duration, gave an average tailing of 0.35 per cent. Cu, and a concentrate carrying 31.6 per cent. insoluble. The capacity per cell seems to be about 15 to 20 tons of slime per day. Sludge acid and sulphuric acid, with no creosote or stove oil, seemed to give as good or better results than when creosote was used.

(*c*) *Round-Table Feed. Mechanical Agitation.*—This test (Period 34), of 27 days' duration, gave an average tailing of 0.30 per cent. Cu, and a concentrate assaying 34.2 per cent. insoluble. The average tonnage per cell was 20 tons per day. The principal oil used was sludge acid kerosene, together with sulphuric acid.

(*d*) *Round-Table Tailing. Mechanical Agitation.*—This test (Period 35), of four days' duration, gave an average tailing of 0.32 per cent. Cu, and a final concentrate of about 60 per cent. insoluble. (The rougher concentrate for the first two days should be included in the average final concentrate.)

(*e*) *Conclusions.*—1. On our slime, air agitation is not as satisfactory as mechanical.

2. The capacity of one standard Callow cell is about 15 to 20 tons of slime per day.

3. The Callow machine produces a clean concentrate but does not give as clean a tailing as the Minerals Separation machine.

4. The Callow machine is more sensitive and requires closer attention than the Minerals Separation machine.

5. The cost of repairs would probably be less on the Callow machine than on the Minerals Separation machine. This cost, however, is comparatively small for either machine.

6. The power required per ton treated in the Callow system is just about the same as that required in the Minerals Separation machine.

In all of these tests the original feed was divided among the Callow rougher cells, operating in parallel. As a rule, there was one cleaner cell operating also. When this was operating the concentrate from the rougher cells went to it, the cleaner making a final concentrate and a middling which was returned to the system. The rougher cells made the final tailing.

2. *Treatment of Mill Tailing after Grinding through 60 Mesh*

(*a*) *Preliminary Tests.*—During the first few shifts the mechanical agitators at the Callow plant were used, but it was soon found that they were not required—that the grinding mill gave sufficient and thorough agitation.

Sludge acid kerosene was the only oil used during this period, and was added ahead of the grinding mill. Sulphuric acid was added ahead of the flotation cells. The tailing for this period averaged 0.10 per cent. Cu, and the concentrate carried an average of 42.2 per cent. insoluble. The pulp was heated just ahead of the flotation cells (Period 36).

(*b*) *Sludge Acid Kerosene Added Ahead of Grinding Mill.*—The flow sheet was modified slightly during Period 37. Instead of sending the entire rough concentrate from the rougher cells to the cleaner cells, only the first half was sent to the cleaner, that coming off the half nearer the tailing end of the machine being returned as middling to the original feed. This resulted in giving the cleaner cell a richer feed and, in turn, enabled the cleaner to make a higher grade concentrate. This period shows some very good results, a tailing assaying 0.11 per cent. Cu and a concentrate carrying 27.9 per cent. insoluble. Here, again, is brought out the splendid agitation and mixing obtained in the grinding mill.

(*c*) *Neutral Circuit. No Acid. Turpentine Added Ahead of Grinding Mill.*—In this test (Period 38), no acid was used. Turpentine was added ahead of the grinder. The tailing averaged 0.16 per cent. Cu, and the concentrate carried 35.1 per cent. insoluble. It should be noted that in this test tonnage treated per cell was only 58.2 tons as against 75.6 tons in the preceding tests.

(*d*) *Neutral Circuit. Same Conditions as under* (*c*) *except Lower Tonnage per Rougher Cell.*—During this period (No. 39) the average tonnage treated per rougher cell was only 39.9 tons. The tailing was high, 0.18 per cent. Cu, and the concentrate low grade, 47.5 per cent. insoluble.

It is evident from the work in this and the preceding period that turpentine will not give us good results in a neutral circuit.

(*e*) *Mixtures of Coal Tar, Creosote and Pine Oil Added Ahead of Grinder. Neutral Circuit.*—The tests during Periods 36 and 37 were made under the supervision of Mr. Bernsen, a representative of Mr. Callow. These

tests show a tailing of 0.13 per cent. Cu, and a low-grade concentrate, — 45.3 per cent. insoluble. The results are not nearly so good as those obtained in the acid circuit.

(*f*) *Same Conditions as under Period* 40, *except the Rougher Cells Operated in Series instead of in Parallel.*—During this test the original feed went to No. 2 rougher and the tailing from No. 2 rougher was fed to No. 1 rougher, the latter making a final tailing. The concentrate from the No. 2 rougher went to the cleaner and that from No. 1 was returned to the feed as middling. The final tailing averaged 0.11 per cent. Cu, but the concentrate was rather low grade, running 38.7 per cent. insoluble.

(*g*) *Conclusions.*—1. The capacity of the standard Callow cell when treating ground mill tailing is about 75 tons per day.

2. No other agitation is required if the reagents can be added ahead of the grinding mill.

3. The use of acid seems to be of considerable advantage.

4. On account of utilizing the grinding mill as an agitator the Callow machine requires less power than the Minerals Separation machine.

5. The Callow machine is more sensitive and requires more attention than the Minerals Separation machine.

The work of the Froment and Towne, and the Fields flotation machines, was found to be unsatisfactory; that on the Anaconda cell was of short duration, no definite results being obtained.

D. Miscellaneous Flotation Tests

1. *Test of Various Kerosene Sludge Acid Oils*

During the experimental flotation work, tests were made on barrel lots of various sludge acid kerosenes, with the following results:

(*a*) *From Union Oil Co. of San Francisco.*—This is the sludge known as M. S. 37, used during practically all of the experimental work. It gives highly satisfactory results. Although exposed to the weather in iron drums during the winter months, it gave no trouble by becoming too viscous.

(*b*) *From Standard Oil Co. of San Francisco.*—This sludge acid gave just as good results in the flotation work as that from the Union Oil Co. This oil did not become viscous when exposed to freezing temperatures.

(*c*) *From Western Oil Co.*—One barrel of this oil was tested during February in the M. S. No. 1 machine when treating round-table feed. This oil did not give as satisfactory results as the two preceding oils. About one-third of the oil was left in the barrel as a solid residue.

(*d*) *From Midwest Refining Co.*—One barrel of this oil was tested during February in the M. S. No. 1 machine when treating round-table feed. The results obtained with this oil were not as good as those obtained when using the first two oils. At least one-half of this oil remained in the

barrel as a solid residue which we could not liquify even by using steam coils.

(e) *From Producers Refining Co.*—We could not test this oil as it was solidified in the barrels and did not become liquid after standing for days at a temperature of 60° F., or better.

2. *Test of Various Creosote Oils*

During the experimental flotation work the following creosote oils were tested with the results shown:

(a) *Creosote from Cleveland-Cliffs Iron Co. of Marquette, Mich.*—This is a creosote obtained as a byproduct in the manufacture of charcoal from hard pine. This is known as M. S. No. 33, the number given to it by the Minerals Separation Co. This is the creosote that was used practically throughout the entire work with highly satisfactory results.

(b) *Creosote from Pensacola Tar & Turpentine Co. of Gull Point, Fla.*—We used a considerable quantity of this oil in our experimental work. It gives fully as good results as the Cleveland-Cliffs creosote. It is made from Georgia pine wood. This oil is known as No. 400 by the Pensacola Co. (see Periods 14C, 19, 20, 21, 22 and 23).

(c) *Creosote from Crichton Pine Products Co. of Alabama.*—A barrel of this oil was tested and gave good results (see Periods 14D and 15A).

(d) *Tar Creosote from Butte Gas Works.*—A test was made using a barrel of tar creosote from the Butte Gas Works, which gave fairly good results. The test was of too short duration to be conclusive, but it is thought that the tar creosote would give satisfactory results (see Period 15B). In testing the Callow machines a mixture of oil containing tar creosote from the Barrett Manufacturing Co. was used with good results (see Periods 40 and 41). In the laboratory tests we have found that the tar creosotes give good results.

(e) *Creosote from the Timber Treating Plant at Rocker.*—This creosote comes from the J. F. Lewis Co. of Moline, Ill. It did not give good results, although the test was too short to be absolutely conclusive. The oil contained a good deal of dirt and sediment (see Period 15E).

3. *Test of Blankets for Callow Cells*

Four blankets for Callow cells were received from Lane & Co., New York City. These blankets were numbered 1,038, 1,039, 1,057, and 1,058. In each case the blankets have a better distribution of air after a few hours' operation, due probably to a readjustment in the weaving after the blanket has been thoroughly wetted.

The blankets were tested, with the following results:

(a) *No.* 1058, *Single Thickness.*—Callow cells treating sand tailing.

The air distribution was uneven and if enough air was turned into chamber to keep sand moving on blanket, then there was violent boiling at points. Blanket not satisfactory for coarse (60-mesh) feed but might work fairly well on slime.

(*b*) *No.* 1,058, *Double Thickness.*—In this test the No. 1,058 blanket was doubled. The feed treated was slime. The air distribution was very good and the results practically the same as the regular Callow blanket supplied by the General Engineering Co. of Salt Lake.

(*c*) *No.* 1,039.—This blanket was tested on slime feed and gave an even distribution for all air pressures up to 5½ lb.

(*d*) *No.* 1,057.—This blanket gave a very uneven distribution of air for all pressures. The pores seemed to close up unevenly when the blanket was wet.

(*e*) *No.* 1,038.—This blanket was tested on slime feed. Gave very good distribution of air for all pressures up to 4 lb. This blanket is thinner than the others and also thinner than the regular Callow blanket and requires less pressure to deliver the same volume of air as the others.

(*f*) *Résumé.*—No. 1,057 would not be satisfactory for our purposes.

No. 1,058, single, would not be satisfactory, except, possibly, on slime feed.

No. 1,058, double, would be satisfactory for either slime or sand, but would probably cost too much.

No. 1,038, entirely satisfactory for slime, and would probably be all right on sand.

4. *Soluble Copper in Flotation Tailing*

Samples of tailing pulp were sent to the laboratory and the soluble copper was determined.

The tailing from the round-table feed contained 0.141 gram of soluble copper per liter of solution. The tailing from the round-table tailing contained 0.084 gram of soluble copper per liter of solution.

5. *Disintegration of Flotation Froth*

In order to break up the froth in the flotation concentrate a saucer-shaped disk 3 ft. in diameter was constructed. This was revolved at a speed of 300 r.p.m. inside a circular tank. The concentrate pulp was fed on to the disk at the center and was thrown out against the sides of the tank. The impact broke up the froth to a great extent. No test was made on the disintegrator.

It was noted early in the experimental work that an elevator made a good froth breaker, the impact of the pulp discharge at the head of the elevator serving to break up the bubbles, to a large extent. A test made at the experimental Callow flotation showed that the rough concentrate pulp was reduced 75 per cent. in bulk in passing through an elevator.

E. Conclusion

The conclusions drawn from the foregoing tests were that the Minerals Separation machine was best adapted for the flotation work at Anaconda. Furthermore, that the most efficient reagents would be sludge acid kerosene, wood creosote, and sulphuric acid.

IV. DESCRIPTION OF REMODELED CONCENTRATOR AS ADAPTED TO FLOTATION

The concentrator at Anaconda, as remodeled for flotation, consists of eight sections, each of 2,000 tons per day capacity, giving a grand total of 15,000 tons per day, allowing for shutdowns, repairs, etc. All sections are alike with the exception of Section 1. In this section, Hancock jigs[1] are used in place of Evans jigs and tube-mills are used in place of Hardinge mills. Fig. 1 shows the flow sheet of Sections 2 through 8.

Milling Division

The ore is fed from the bins to a 2-in. round-hole shaking screen, the oversize going to a 12 by 24-in. Blake crusher. The product from this crusher is delivered to a 2-in. round-hole trommel, the oversize of which is sent to two 8 by 20-in. Blake crushers. The product from these crushers, together with the undersize from the 2-in. screens, is elevated and passed through 1-in. round-hole trommels. The oversize from this is treated in coarse Harz jigs, making a middling and a concentrate; the undersize is passed through ⅜-in. trommels, the oversize being treated in fine Harz jigs making a concentrate and a middling. All sections are alike up to this point. In Section 1, the undersize from the ⅜-in. trommel is screened on 1½ by 12-mm. trommels, the ucndersize going to the Anaconda classifiers and the oversize to the Hanock jigs. The treatment of the products from this point is the same in all sections, except that Section 1 uses tube-mills in place of Hardinge mills for grinding, as noted previously. The undersize from the ⅜-in. trommel is screened through 4-mm. trommels, the oversize from these going to the double compound Evans jigs and the undersize going to 1½ by 12-mm. trommels. The undersize from these trommels goes to the Anaconda classifiers, the oversize to double compound Evans jigs. The two sets of Evans jigs make a concentrate which goes to the dewatering bins and a middling which is ground for further treatment.

The concentrate from the coarse Harz jigs is dewatered and conveyed to bins. The middling is screened on a dewatering screen, the undersize together with the hutch product from the coarse Harz jigs going

[1] For comparative data on Hancock and Evans jigs, see *Trans.*, vol. 46, p. 212 (1913)

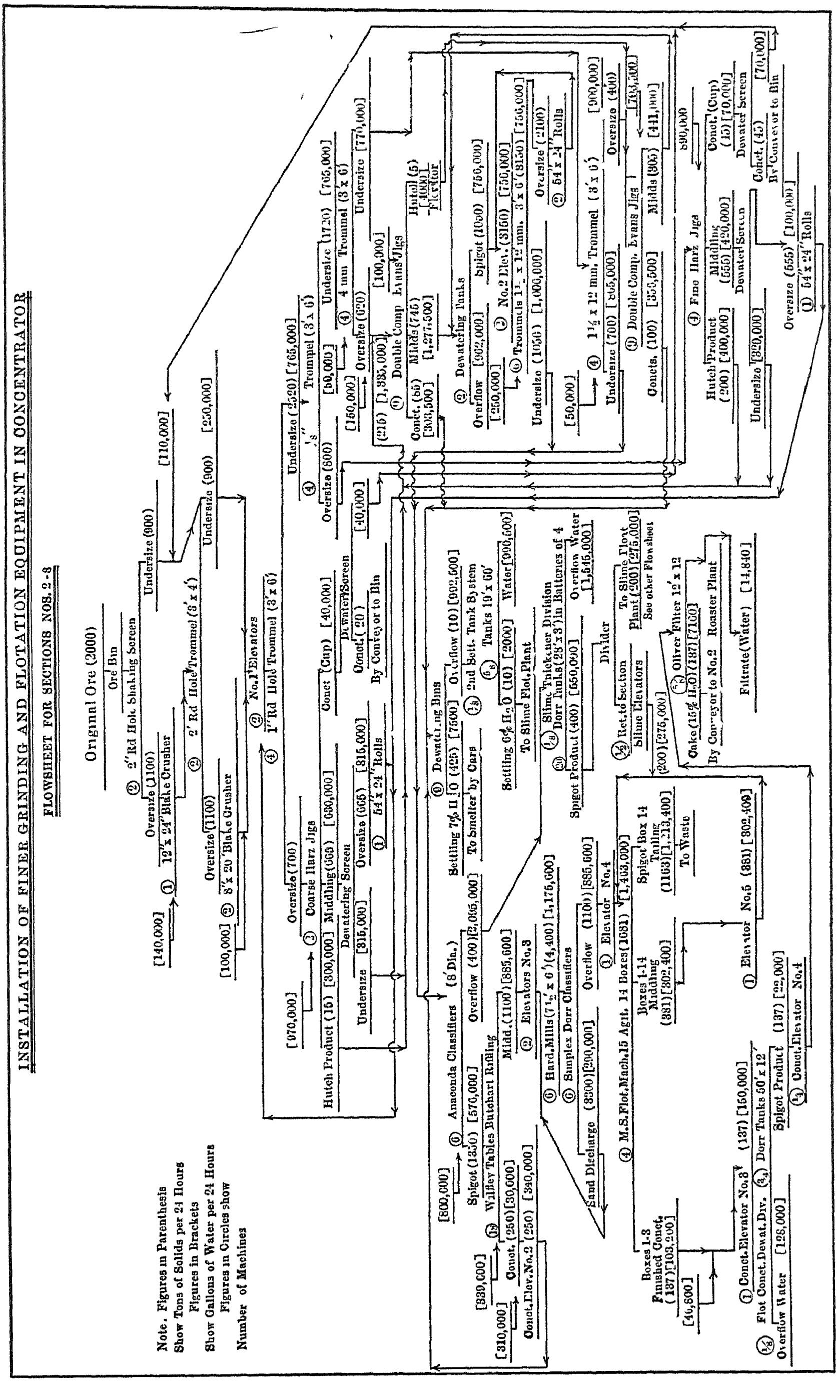

INSTALLATION OF FINER GRINDING AND FLOTATION EQUIPMENT IN CONCENTRATOR
FLOWSHEET FOR SECTIONS NOS. 2-8
Note. Figures in Parenthesis Show Tons of Solids per 24 Hours
Figures in Brackets Show Gallons of Water per 24 Hours
Figures in Circles show Number of Machines
Original Ore (2000)
Ore Bin
2" Rd Hol. Shaking Screen
12" x 24" Blake Crusher
2' Rd Hole Trommel (3' x 4')
8" x 20" Blake Crusher
No. 1 Elevators
1" Rd Hole Trommel (3' x 6')
Coarse Harz Jigs
Dewatering Screen
54" x 24" Rolls
Anaconda Classifiers (8' Dia.)
Wilfley Tables Butchart Riffling
Elevators No. 3
Hard. Mills (7½' x 6')
Simplex Dorr Classifiers
M.S. Flot. Mach. 15 Agit. 14 Boxes
Elevator No. 4
Elevator No. 5
To Waste
Dewatering Bins
To Smelter by Cars
2nd Sett. Tank System
To Slime Flot. Plant
Slime Thickener Division
Divider
To Slime Float Plant
See other Flowsheet
Oliver Filter 12' x 12
Roaster Plant
Slime Elevators
Dewatering Tanks
Double Comp. Evans Jigs
Fine Harz Jigs
Dewater Screen
By Conveyor to Bin

FIG. 1.

to the Evans jigs. The oversize is passed through rolls, 54 by 24 in., and thence back into the system ahead of the 1-in. round-hole trommels. The concentrate from the fine Harz jigs is sent to the bins. The middling is screened through a dewatering screen, the oversize going to 54 by 24-in. rolls and then back into the system ahead of the 1-in. round-hole trommels. The undersize of the dewatering screen together with the hutch discharge of the fine Harz jigs goes to the Evans jigs.

The concentrate from the Evans jigs is dewatered in bins to about 7 per cent. moisture, and sent to the smelter. The jig concentrate assays

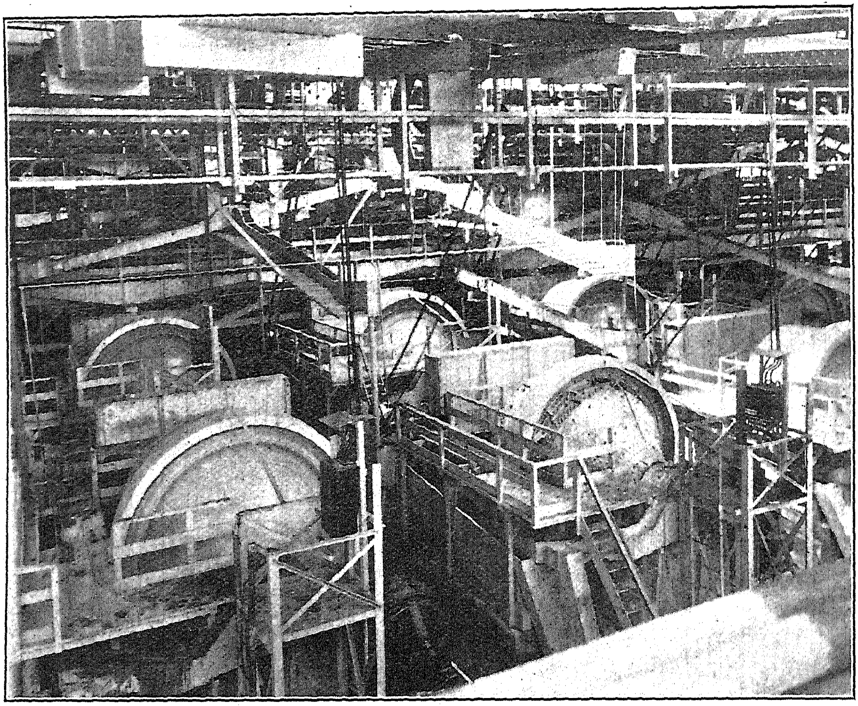


FIG. 2.—GRINDING DIVISION.

about 15 per cent. insoluble and 8 per cent. copper. The middling, together with the hutch product, is dewatered in tanks and screened through 1½ by 12-mm. trommels, the undersize from which goes to the Anaconda classifiers,[2] the oversize through 54 by 24-in. rolls, and back to the 1½ by 12-mm. trommels.

The spigot from the Anaconda classifier is treated on 18 Wilfley tables, fitted with Butchart riffling, making a concentrate and a middling. These tables make a concentrate assaying 25 per cent. insoluble and a middling assaying 0.9 per cent. Cu. The concentrate is sent to the

[2] For description of Anaconda Classifier, see *Trans.*, vol. 46, p. 277 (1913).

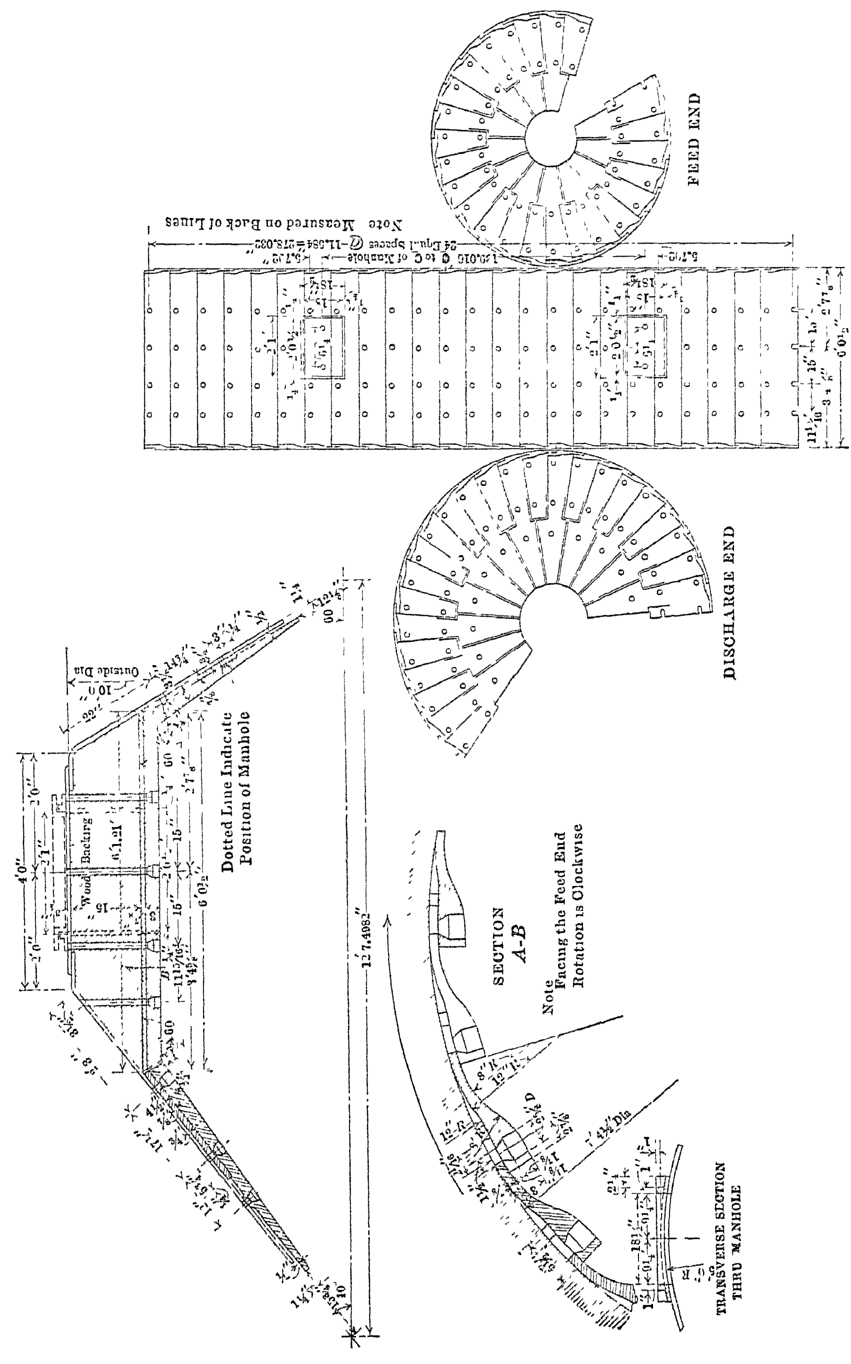
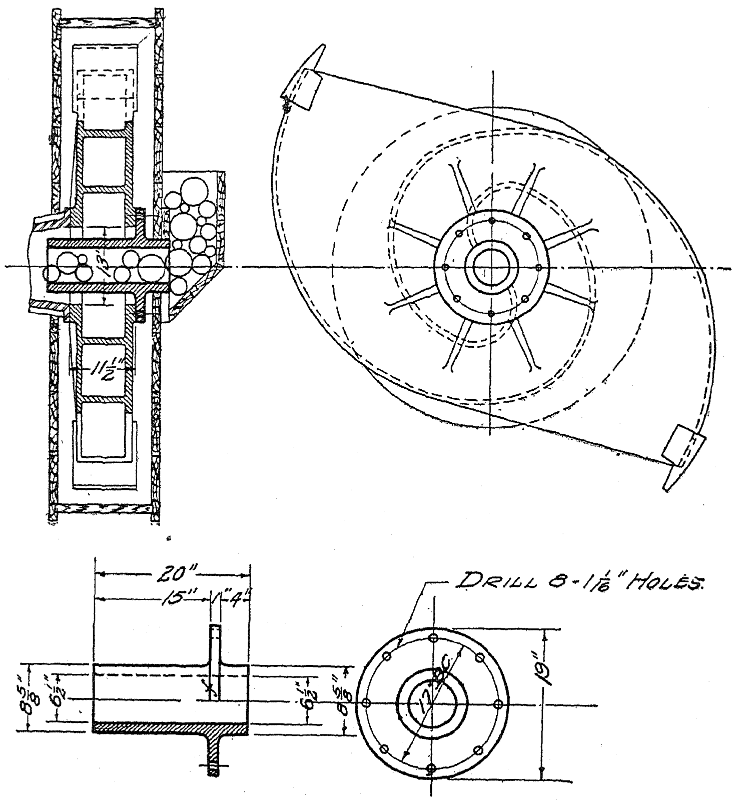

FEED END
DISCHARGE END
Note Measured on Back of Lines
Dotted Line Indicate Position of Manhole
Wood Backing
Outside Dia
SECTION A-B
Note Facing the Feed End Rotation is Clockwise
TRANSVERSE SECTION THRU MANHOLE

FIG. 3.

dewatering bins, together with the fine jig concentrate, and the middling is sent to the 10 by 4-ft. Hardinge mills. The overflow from the Anaconda classifiers is sent to the slime thickener division, consisting of 28 by 3-ft. Dorr tanks.[3] The spigot product from these tanks is divided; about one-half is returned to the section and the remainder is sent to the slime plant.

The product from the Hardinge mills (Fig. 2) is treated in six simplex



Patent applied for.

Fig. 4.—Spout for Feeding Pebbles or Balls into Hardinge or Tube Mills.

Dorr classifiers—one classifier to each mill—the overflow going to the flotation division and the classifier sand being returned to the mill.

At the time it was first decided to remodel the concentrator, it was not definitely known whether pebbles or steel balls would be used for grinding. To provide for this uncertainty a compromise was effected. The mills were made 10 by 4 ft. and built sufficiently strong for steel balls in case balls were used. Each mill was equipped with a 225-hp. motor directly connected through a flexible coupling. The mill filled

[3] For description of this thickener plant, see *Trans.*, vol. 49, p. 470 (1914).

with pebbles takes from 95 to 115 hp. to operate. In case steel balls were used it was planned to put in a false wood lining back of the steel lining in the cylindrical part of the mill to reduce the effective diameter of the mill.

This latter plan was finally adopted, and the Hardinge mills will be equipped with the false wood lining, 15 in. thick, in the cylindrical part of the mill, and a Cascade steel lining. With this form of lining, the mill is virtually 7½ by 6 ft. and requires about 225 hp. when loaded with steel balls.

The drawing (Fig. 3) gives the details of the lining. This lining was designed by the American Manganese Steel Co.[4] At first the pebbles, and later the balls, were fed to the mills through the feed scoop. This method of introducing the grinding medium into the mill gave considerable trouble, due to the breaking of the feed boxes caused by the jamming of a pebble or ball between the revolving scoop and the feed box. We tried to obviate this difficulty by various changes in the amount of clearance left between the scoop and the box, but without success. In our particular case this trouble was aggravated by the fact that we had to use 7 ft. diameter scoops, in order to lift back into the mill the sand discharged by the Dorr classifier. Finally a method was tried of feeding the pebbles, or balls through a spout passing through the center of the feed scoop. This device has worked splendidly and all of our mills have since been equipped with it (see Fig. 4).

Flotation Division

The flotation division consists of four Minerals Separation machines, each having 15 agitators 3 ft. square, and 14 spitzkästen or floating compartments. The agitators for the Minerals Separation machines are of gun metal and are driven by bevel gears from a line shaft, the direction of rotation of the agitators alternating.

The machines are made of California red wood; the agitator boxes are further lined with hard maple extending about 18 in. from the bottom of the box.

Each machine has an individual drive, power being supplied to the line shaft by a 150-hp. motor running at 385 r.p.m. The speed of the agitators is 225 r.p.m. and as the impellers are 18 in. in diameter the peripheral speed is about 1,060 ft. per minute.

Each machine makes three products, a concentrate, which goes to the dewatering division, a middling which is returned to the head of the machine, and a tailing which goes to waste. The concentrate is taken from the first three to five spitzkästen and the middling from the last nine to eleven. A portion of the pulp is overflowed from the last three

[4] A detailed description of these mills, together with grinding data and Dorr classifier data, will be published in a subsequent paper.

spitzkästen together with the froth. About 6 to 8 lb. of 50°Bé. H_2SO_4 per ton of flotation feed is used together with 2 to 3 lb. of kerosene sludge acid and ½ to 1 lb. of crude wood creosote. A portion of the wood creosote is added ahead of the Hardinge mill (about 0.03 to 0.05 lb. per ton of feed) and the remainder is added in the sixth agitating compartment. The sulphuric acid and sludge acid are added at the head of the machine. The pulp is heated to from 60° to 70° F., by passing live steam into it at the head of the machine. Three machines are used for treating sand and the fourth for treating current slime from the upper portion of the mill. Each machine has a capacity of about 400 tons per day on sand and 175 tons on slime.

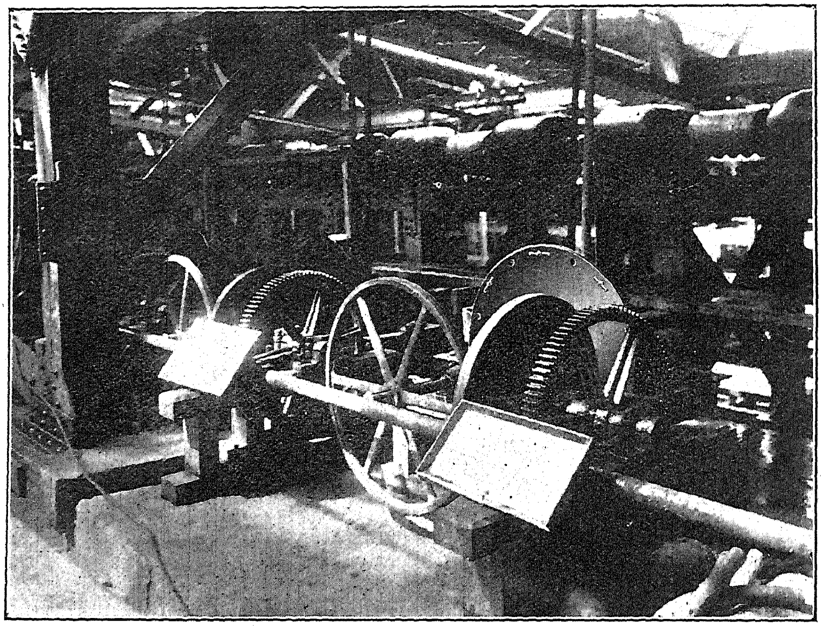


Fig. 5.—Automatic Reagent Feeders.

The method of adding the oil and acid is rather unique. The mechanism consists of a revolving disk to which are attached, around the circumference, a number of cups. This disk is set vertically so that its lower edge dips into a pan of acid or oil. As the cups come around they are filled and later discharge their contents into a suitable launder leading to the flotation machine. The disk is driven by the friction of a wheel against another disk attached to the main drive. The wheel is run at constant speed and by varying the point of contact between wheel and disk any speed desired can be given to the main disk and thus the amount of oil or acid added can be regulated. In addition to the speed regulation, the amount of oil or acid fed may be varied by adding or removing cups

or by changing the size of the cups. A photograph of the machine is shown in Fig. 5.

At present (December, 1915) seven sections are operating on the new flow sheet, and the whole mill will be remodeled not later than Jan. 15, 1916. The sections are being remodeled one at a time. All the work is being done by the company's engineering force. Things have been so well organized and systematized that it requires less than 30 days to tear out the old section and install the new equipment, ready for operation.

The following table gives metallurgical data concerning the operation of the mill. Line 1 gives the monthly assay of second-class ore for October, which is the feed to the mill. Line 2 gives the monthly assay of the flotation tailing from the remodeled sections for October. The feed to the flotation machines during the month consisted of reground sand tailing and a portion of the thickened mill slime which was returned to the mill and treated in the fourth machine in each remodeled section. The tailing assay shown below, 0.13 per cent. Cu, is for the total sand and slime tailing produced. The sand tailing alone averaged 0.10 per cent. Cu and the slime tailing averaged about 0.25 per cent. Cu.

	Per Cent. Cu	Per Cent SiO_2	Per Cent. Al_2O_3	Per Cent. Fe	Per Cent. S	Per Cent CaO	Ounces per Ton	
							Ag	Au
Feed to concentrator	2.85	59.1	9.4	10.6	11.8	0.6	2.00	0.0070
Total concentrator tailing (flotation)	0 13	81.6	11.3	1 0	0.5	0.5	0.10	0.0005

No complete analyses of the flotation concentrate are available, as the slime flotation plant is not in operation yet and the round-table concentrate is being mixed with the flotation concentrate from the mill.

Following is an estimate of the power consumption per 2,000-ton section, including its proportion of slime treatment.

	Power Consumption per Section, Horsepower
Feeding and coarse crushing	125
Roll crushing	220
Jigging	60
Screening	37
Conveying and elevating	237
Table concentration	25
Fine grinding	655
Flotation of sand and slime in mill	426
Dewatering mill slime pulp	3
Flotation of slime in slime division	109
Dewatering flotation concentrate	25
Sampling and assaying	1
Total	1,923
Horsepower per ton of ore	0.96

III. DESCRIPTION OF SLIME-FLOTATION PLANT

Because of lack of space in the mill an auxiliary plant had to be installed to handle the extra slime. This plant consists of 20 Minerals Separation machines of the same type and size as those used in the concentrator, and five 50 by 12-ft. Dorr tanks for dewatering the concentrate (Figs. 6, 7 and 8). The plant is designed to treat about 2,000 tons of current slime and 1,000 tons of pond slime. This plant will be operating by Dec. 15, 1915.

What this plant may be expected to do, so far as the treatment of current slime is concerned, may be judged from the results obtained in the experimental slime machine, which has been operating regularly for more than a year.

The following table gives the analysis of the current slime and the assay of the composite tailing sample for the month of June. These are typical of the results which may be expected in the large plant.

Operation of Experimental Slime-Flotation Plant

Per Cent	Feed (Current Slime)	Tailing	Concentrate
Cu	2.10	0.27	12.0
SiO_2	61.00	67.70	20.0
FeO	4 10	2.70	28.0
S	4.40	1.10	
Al_2O_3	19.00	18.30	
CaO	0.60	0.70	
Ag[1]	1.80	0 20	
Au[1]	0.005	0 001	

[1] Ounce per ton.

The treatment of the pond slime offers a few more difficulties than that of the current slime. This slime has been exposed to the weather for a number of years so that the copper is partially oxidized, consequently that portion probably cannot be floated. Some results from the flotation plant at Great Falls, covering a week's operation, are tabulated below. This character of material has been treated for some time at the Great Falls plant, and the results should be a good criterion of what may be expected here in the treatment of dump slime.

Operation of Slime-Flotation Plant at Great Falls

	Per Cent.
Recovery based on concentrate	78.90
Recovery based on tailing	80.90
Average total Cu in tailing	0.53
Average soluble Cu in tailing	0.13
Average sulphide Cu in tailing	0.40
Average total Cu in feed	2.52

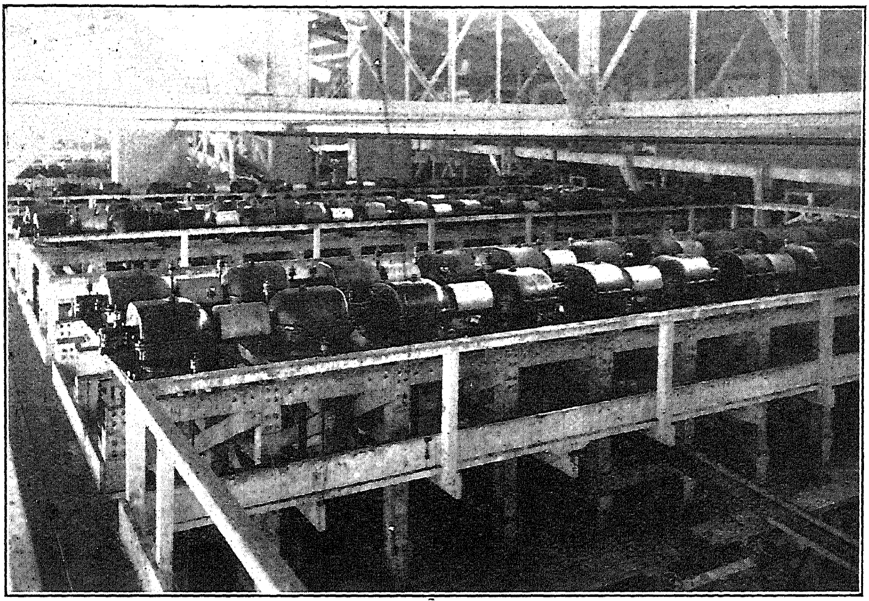


FIG. 6.—MINERALS SEPARATION MACHINES IN SLIME-FLOTATION PLANT.

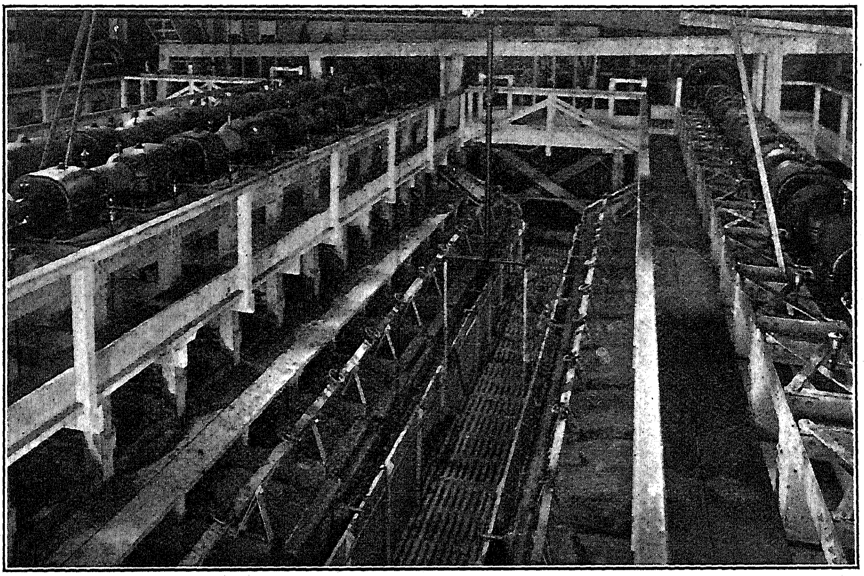


FIG. 7.—MINERALS SEPARATION MACHINES IN SLIME-FLOTATION PLANT. (MACHINE ON LEFT IS NOT IN OPERATION.)

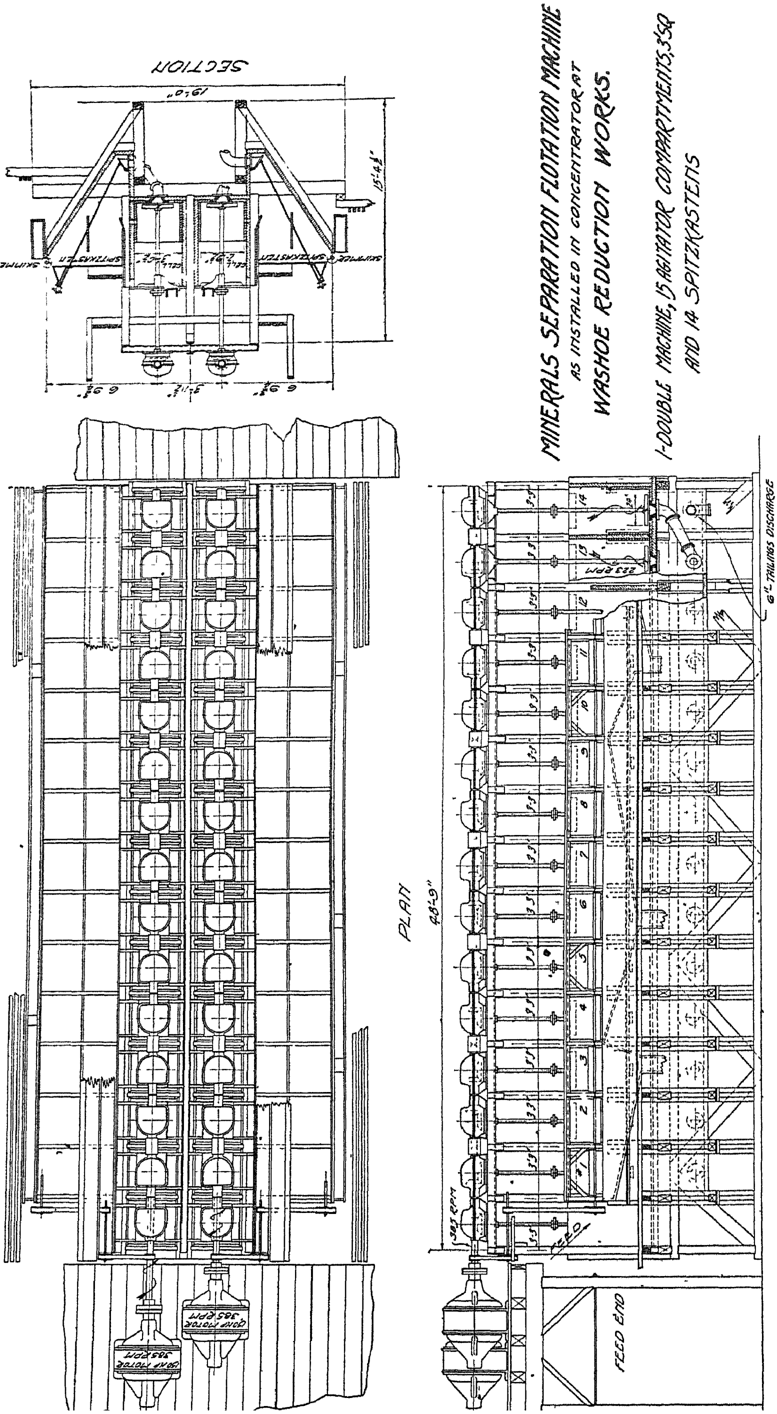

MINERALS SEPARATION FLOTATION MACHINE
AS INSTALLED IN CONCENTRATOR AT
WASHOE REDUCTION WORKS.
1-DOUBLE MACHINE, 15 AGITATOR COMPARTMENTS, 3'SQ
AND 14 SPITZKASTENS
SECTION
19'-0"
15'-4½"
SKIMMER
SPITZKASTEN
PLAN
48'-9"
ELEVATION
FEED END
FEED
6" TAILINGS DISCHARGE
223 RPM

FIG. 8.

At Great Falls, one double Minerals Separation machine of the same design and size as used at Anaconda is treating from 300 to 350 tons of dump slime daily.

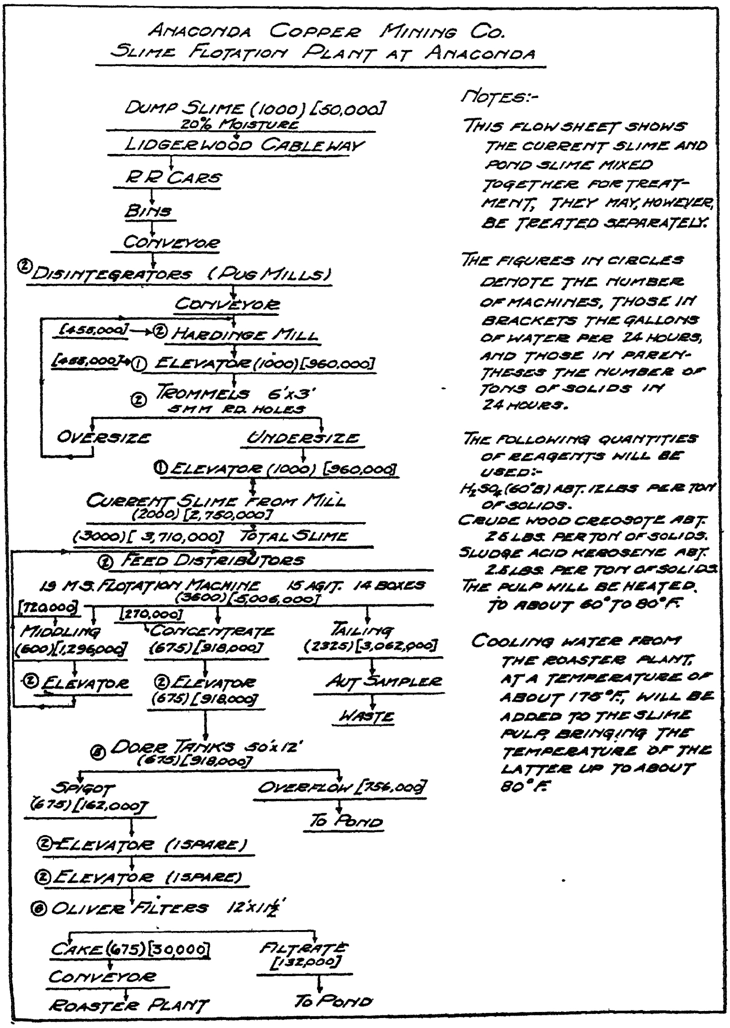


FIG. 9.—FLOW SHEET OF SLIME-FLOTATION PLANT AT ANACONDA, MONT.

SULPHIDIZATION EXPERIMENTS AT GREAT FALLS

Experiments covering a period of five days, in which five tons of sodium sulphide were consumed, were made at the Great Falls plant, to determine the feasibility of sulphidizing the oxidized copper prior to flotation. Sodium sulphide, in solution, was added at the pug mill

together with the dump slime. No difficulty was found in sulphidizing practically all of the copper which went into solution, but it was found that all oxidized copper was not dissolved during its passage through the pug mill and flotation machine.

Moreover, on the addition of sodium sulphide, the froth immediately broke up—a large part of the natural sulphides no longer floating, and the tailing assay showing a marked increase, although the precipitated sulphides floated. Hardly any froth was formed, probably because of the colloidal state of the precipitated sulphides rather than the result of any deleterious action on the oils used of the sodium sulphide, or the sodium sulphate formed.

The cause of the breaking of the froth was not clear. It may have

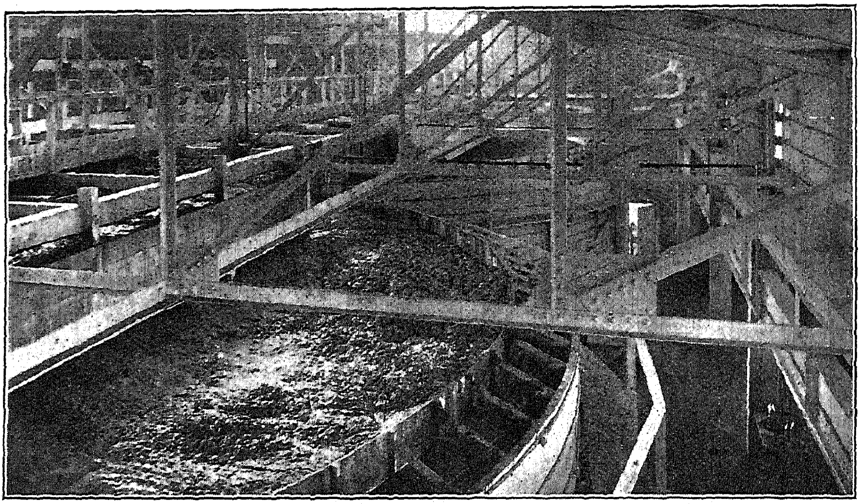


FIG. 10.—DORR TANKS IN SLIME-FLOTATION CONCENTRATE-DEWATERING DIVISION.

been due to the sodium sulphate formed, or, what seems more probable, to some impurities in the sodium sulphide, the latter being only about 50 per cent. pure. This effect was very general, the froth on the second machine breaking up, although no sodium sulphide was added to it, except through a return middling elevator which was common to both machines. Fig. 9 shows the flow sheet of the slime-flotation plant at Anaconda.

IV. DESCRIPTION OF FLOTATION CONCENTRATE DEWATERING PLANT

The dewatering of the flotation concentrate is done in six 50 by 12-ft. Dorr thickeners. Five tanks of the same size have been provided for the slime plant. The pulp is delivered to a baffle box about 5 ft.

square in the center of the tank, and extending down to within a few inches of the rake arms. Surrounding this is another baffle about 15 ft. square and extending about 18 in. below the surface of the water (Fig. 10). These baffles catch a large portion of the froth which is there broken up by means of a water spray. The capacity of these tanks when treating flotation concentrate is from 200,000 to 250,000 gal. of pulp per 24 hr., although they will probably not be required to handle more than 200,000 gal. Operating even at this capacity, there is a small amount of finely divided material that will not settle. Therefore, the overflow from these tanks is run to a slime pond and the solids collected for future treatment. The capacity of these same tanks when treating round-table concentrate is about 1,000,000 gal. of pulp per 24 hr.

Some experiments were made to increase the capacity of the tanks by the use of glue. Tests made in a beaker seemed to indicate that glue would greatly aid the settling, but this did not prove to be the case in practice. The glue caused the colloidal material to coagulate, but it did not actually increase the capacity of the settling tanks to any great extent.

The density of the pulp delivered to the tanks averages from 18 to 20 per cent. solids, and the spigot averages about 60 per cent. solids. The spigot product is further dewatered in Oliver filters 11½ ft. diameter by 12 ft. face. There will be eleven of these, each capable of handling 150 tons per 24 hr. and making a cake containing about 15 per cent. moisture. The cake from the filters is delivered to a belt conveyor and is carried with the fine mill concentrate to the new roaster plant.[5] In this way a fairly good mixing of the concentrates will be obtained.

[5] The new roaster plant consists of 28 25-ft. Wedge furnaces. This will about double the roaster capacity of the plant.

TABLE I.—*Summary of Flotation Tests—Minerals Separation Machine No. 1 Treatment of Slime: Round-Table Feed and Tailing*

Period	Feed				Concentrate Assay				Tailing Assay, Per Cent Cu	Per Cent Cu Recovered	Temperature, °F.			Reagents Used, Pounds per Ton						P. T. & T. Co. 400
	Tons per 24 Hr.	Assay, Per Cent Cu	Density, Per Cent. Solids	Hours in Operation	Per Cent Cu	Per Cent Insol.	Per Cent FeO	Ag Oz per Ton			Feed Pulp	Pulp in Machine	Difference	Sludge Acid	Wood Creosote	Stove Oil	H_2SO_4	Turpentine	Tar Creosote	
1	Preliminary																			
2	61.0	2.04	13.7	120.0	8.2	54.7	13.7	...	0.31	89.3	54	89	35	3.8	2.4	0.36	12.6			
3	64.2	2.07	14.9	72.0	7.2	62.0	10.8	6.0	0.38	86.0	51	51	0	3.6	2.3	0.29	13.3			
4	56.2	2.08	13.6	48.0	8.3	51.1	15.6	6.7	0.27	90.4	53	125	72	3.9	2.8	0.33	12.6			
5	52.2	1.16	11.7	56.0	6.6	66.1	7.8	5.8	0.33	75.5	54	54	0	3.5	3.8	0.37	15.2			
6	48.2	1.08	11.1	72.0	7.2	62.1	8.9	6.8	0.31	74.3	54	106	52	3.9	4.2	0.29	11.8			
7	63.8	2.24	22.5	40.0	9.2	52.9	12.8	7.8	0.28	90.3	53	96	43	2.7	2.4	0.33	12.5			
8	104.8	2.19	24.5	88.0	8.9	57.5	11.2	6.45	0.63	76.6	53	96	43	3.1	2.4	0.32	13.9*a*			
9	94.7	2.25	22.7	120.0	8.7	60.5	9.2	6.45	1.34	47.5	54	54	0	2.0	2.2	0.21	16.4			
10	60.4	2.20	14.5	96.0	10.1	52.6	12.4	7.9	0.50	82.0	48	48	0	3.3	2.9	0.21	16.5			
11	58.2	2.39	14.5	144.0	9.2	53.3	12.8	6.8	0.33	89.5	50	81	31	3.5	3.4	0.42	12.7			
12	56.1	1.11	18.1	24.0	5.9	71.8	5.2	4.4	0.53	57.8	52	72	20	3.7	3.7	0.28	16.1			
13A	60.6	1.11	13.5	184.0	5.8	71.3	5.6	4.2	0.28	79.1	52	80	28	3.7	3.5	0.30	16.8			
13B	51.6	1.35	11.1	32.0	2.0	85.0	2.5	1.0	1.04	47.8	51	80	29	5.5	3.1	0.27	00.0			
14A	31.7	1.17	5.6	48.0	6.8	66.2	7.4	4.6	0.30	77.8	50	80	30	5.1	4.3	0.32	18.9			
14B	42.6	1.17	6.8	24.0	5.9	68.0	7.2	4.6	0.24	82.7	50	71	21	4.4	......	0.37	21.0	3.7		
14C	40.4	1.12	6.7	80.0	6.1	67.2	8.3	4.8	0.23	82.5	52	80	28	4.4	4.0	0.28	18.5			
14D	40.0	1.13	6.5	16.0	9.8	54.3	11.5	6.6	0.32	....	52	81	29	5.0	4.2	0.73	20.1			
15A	42.7	2.35	9.4	40.0	9.3	47.2	17.0	6.7	0.23	92.5	48	82	34	4.3	4.5	0.33	20.7			
15B	48.0	2.51	8.0	32.0	13.4	33.6	20.9	8.8	0.26	91.5	48	78	30	5.4	.... ..	0.34	20.0	....	4.7	
15C	49.0	2.52	8.8	80.0	10.8	44.2	18.4	8.3	0.25	92.2	45	71	26	3.8	......	0.41	19.3	3.3		
15D	49.4	2.25	9.7	40.0	11.0	40.7	19.3	8.1	0.38	86.0	46	70	24	9.0	....	0.33	19.3			
15E	50.2	2.52	9.3	40.0	14.3	28.9	23.6	11.4	0.48	83.8	44	70	26	4.0	.	0.49	18.6	...	4.5	
16A	49.0	2.55	11.5	248.0	9.7	48.8	16.0	7.7	0.27	92.1	44	68	24	3.2	4.7	0.46	20.9			
16B	55.2	2.58	13.7	24.0	10.6	48.5	15.6	9.2	0.56	83.0	45	66	21	5.9	4.6	0.31	23.2			
17	40.0	1.10	5.8	576.0	7.5	63.0	9.3	5.9	0.22	82.4	43	69	26	4.7	4.8	0.30	16.7			
18	42.0	1.11	5.8	334.9	7.0	65.9	7.7	4.7	0.26	79.5	40	70	30	4.1	4.54	0.19	19.6			
19	90.0	2.57	15.7	236.4	14.8	31.3	22.1	11.2	0.43	85.8	39	72	33	2.54	1.38	0.14	13.1	.. ...	....	1.61
20	60.2	2.74	15.3	211.3	12.0	38.3	20.6	8.8	0.33	90.5	39	70	31	3.27	...	0.26	15.6	..	......	1.73*b*
21	40.9	2.48	15.1	313.1	10.1	43.7	18.8	6.9	0.25	92.3	40	76	36	6.15	......	0.29	19.5	. ..	....	3.60
22	88.2	2.54	15.9	417.7	11.9	39.4	20.1	8.4	0.25	92.3	41	74	33	2.99	......	0.13	13.5	...	.....	1.88
23	89.9	2.60	15.4	283.7	12.3	36.4	20.8	9.3	0.27	91.6	43	74	31	3.27	... ..	0.18	14.9	.	.. ..	1.52
30*c*	93.3	2.55	10.7	336.0	8.3	55.2	13.3	5.8	0.41	99.3	44	65	21	3.7	3.4	0.71	21.1			
31*c*	45.1	1.10	5.7	720.0	4.5	74.3	5.7	.. ..	0.27	80.3	44	65	21	3.9	3.3	0.34	18.7			

NOTE.—This table gives the average of the results of each period.
a During this period about 1.8 lb. of fuel oil per ton of feed was used with the other oils.
b During two days of this test Midwest Refining Co. sludge acid was used along with the other reagents at the rate of 3.69 lb. per ton, and on another day Western Oil Co. sludge acid at the rate of 1.04 lb. per ton No improvement found in either instance.
c These tests were made in M. S. Machine No. 2.

TABLE IA.—*Summary of Tests in Minerals Separation Machine No. 1*
Treatment of Sand Tailing from Mill, Ground through 0.25 Mm.

Period	Feed				Concentrate Assay				Tailing Assay, Per Cent Cu	Per Cent Cu Recovered	Temperature, °F.			Reagents Used, Pounds per Ton						P. T. & T. Co 400
	Tons per 24 Hr.	Assay, per Cent. Cu	Density, Per Cent Solids	Hours in Operation	Per Cent. Cu	Per Cent Insol	Per Cent FeO	Ag Oz per Ton			Feed Pulp	Pulp in Machine	Difference	Sludge Acid	Wood Creosote	Stove Oil	H_2SO_4	Turpentine M.S. 14	Tar Creosote	
24	Preliminary																			
25	125.0	0.62	22.3	55 8	6.35	40 1	26 8	4 2	0.07	93.0	40	69	29	2.5	.....	....	7 6			
26	186 7	0 87	30.9	472.6	7.82	26.9	32.5	5.4	0 11	88 6	43	69	26	3 1	0.7	....	4 9	. .	....	0 3[1]
27	124 1	0·67	26.1	120 5	5.42	42.2	25.4	4.2	0.12	84.0	42	72	30	......	....	0 3	1.3	3 9		

[1] Average of four days' test. Results not improved by this oil. For two days ½ lb. of Standard Oil Co. Sludge Acid Kerosene per ton of feed was used along with the P. T. & T. Co. No. 400. No marked improvement noted. Wood creosote used for only six days. Tailing assayed about 0 09 during this time.

TABLE II.—*Treatment of Round-Table Feed by Flotation Process. Minerals Separation Machine No. 2A*

Date, 1915	Rate per 24 Hr.		Assay Feed, Per Cent. Cu	Temperature of Circuit, °F.	Tailing Assay, Per Cent Cu	Concentrate, Per Cent.			Reagents, Lb. per Ton			
	Gallons Pulp	Tons Solids				Cu	Insoluble	FeO	H_2SO_4	M S 37	P T & T Co 400	M. S 8
2–27	58,900	39.5	2.59	90.9	0 86	8 3	54 4	14 3	26 3	6 9	2 3	0 15
2–28	40,200	29.8	2 50	87.2	0.73	9 8	36 9	27.1	30.1	7.0	3.2	0.01
3–1	44,600	29.7	2.50	86.1	0.93	10 0	46 4	18 3	26.1	7.3	5.3	0.50
3–2	49,300	34 6	2.55	92.6	0.50	5.9	62 2	11.9	20 9	6.5	2 7	0 20
3–3	52,200	36.4	2 62	90.7	0.47	6 0	60 5	12.6	22 3	6 1	1.3	0.07
3–4	55,100	39.7	2 54	86 0	0 61	7.0	58 0	13 0	17 6	5 1	2 4	0 03
Average....	50,050	34.9	2.55	88.9	9 68	7 8	53 1	16 2	20 5	6.5	2.9	0 16

The machine was in good running order, mechanically, on Feb. 27, 1915. There seemed to be too much agitation at first, so No 4 impeller was dropped off shaft. Machine ran this way until Mar. 3, when the No. 2 impeller was dropped. Air was blown in under each impeller. With first and third impellers operating, the input to the motor was 32 hp.

Table III.—*Summary of Flotation Tests—Callow Pneumatic Machine Treatment of Slime*

Period	Hours in Operation	Number of Rougher Cells Operating	Air Pressure, Pounds per Square Inch	Feed				Temperature of Pulp in Circulation	Reagents Used, Lb. per Ton							Final Tailing, Per Cent Cu	Final Concentrate			Middling, Per Cent Cu,	Rougher Concentrate			Per Cent. Cu Recovered
				Per Cent. on 80 Mesh	Density, Per Cent Solids	Assay, Per Cent. Cu	Tons per Rougher Cell		H_2SO_4	Sludge Acid	Crude Wood Creosote	Stove Oil	Turpentine	Fuel Oil	Pine Tar		Per Cent Cu	Per Cent. Insol	Per Cent. FeO		Per Cent. Cu	Per Cent. Insol	Per Cent FeO	
32	Preliminary																							
33*a*	246.5	2.8	4.52	......	14.7	2.57	15 0	72 4	18 4	5 75	3.04*b*	0.51*b*	....	...	...	0 35	13.7	31 6	23.6	1.71	6 7	54 4	13 8	88 6
34*c*	500.1	2.0	4.00	......	15.2	2.57	20 2	71 1	12.4	3.82	3.13	0.35	....	0 24	...	0 30	12.0	34 2	23.4	1 69	6 0	56 8	13.3	90.6
35*c*	42 8	3 3	4.00	......	5.6	1.66	11 4	68.8	18.5	9.05	.....	.....	....	0 71	....	0 32	12 7	53.9	10.0	......	6 0	64 8	6 5	83.3
Treatment of Mill Tailing																								
36	249.0	2.0	4.00	13.4	27 5	0 64	108.5	64.5	3 3	2.4	....	.....	....	. ..	....	0.10	6.53	42.2	25.0	0 52	2 9	79 1	7.4	
37	120.8	2.0	4.00	6 5	27.1	0.77	75.6	72 6	5.3	2.5	....	.....	....	....	....	0.11	7 81	27 9	32.4*d*	1 25	3.8			
38	40.0	1.6	3.90	6.9	22.1	0.65	58 2	69.9	0.0	0.0	.. ..	.....	2.7	...	....	0.16	6 23	35 1	28 8*e*	1.18	3 2			
39	76.7	......	2.60	.....	22 0	0 72	39.9	67.4	....	...	....	....	3.09	..	...	0 18	5 6	47.5	32 2					

a Air agitation used throughout this period.
b These oils were used only during first five days.
c Mechanical agitation used.
e Concentrate assayed 5.2 oz. silver per ton.
d Concentrate assayed 4 8 oz silver per ton.

Discussion

O. C. Ralston, Salt Lake City, Utah.—I have merely glanced over this paper, consequently I am hardly in a position to discuss it intelligently. There is one thing, however, that is of interest, that is, the claim that the oil mixture which they have chosen is the best adapted to the work at hand. I would like to ask Mr. Mathewson if they still feel that that is the case—if that really does give the best grade of concentration and the best extraction for the same money, compared to any particular oil mixture which they could get.

E. P. Mathewson, Anaconda, Mont.—We are still using that mixture, and we consider it the most economical mixture used.

David Cole, El Paso, Texas.—I would like to ask Mr. Ralston if there are some other mixtures that would be better?

E. P. Mathewson.—We are open to suggestions.

O. C. Ralston.—My own paper on oils* calls attention to the fact that certain hardwood oils, especially the tars, are now being burned or wasted. A great deal of hardwood tar is available as low as 4 or 5 c. per gallon, that being its value as fuel under the boilers. As a matter of fact, a great deal of testing has been done with only the commercial products that are as a usual rule sold on the market. Very little hardwood tar is sold on the market. In asking a great many people who have tested flotation oils, I have not heard hardwood tar mentioned very often. I think I would add to that that the hardwood tars can probably be bought in quantity comparable with the sludge acid at Anaconda, and should contain a greater proportion of the frothing constituent, and at a price fairly commensurate with what they pay for the sludge acid.

E. P. Mathewson.—I would like to make a correction in one statement I made, that there has been no change made in the mixture. This change has been made: We found that the amount of wood creosote in treating the sand from tables was extremely small, and we tried some experiments on a large scale, dropping it out and using simply the sludge acid and sulphuric acid. We found that this gave practically as good results as wood creosote. We use wood creosote in treating the slimes. We find it necessary in that operation.

David Cole.—I would like to ask, what is sludge acid?

E. P. Mathewson.—Sludge acid is refuse from the refining of oil. It contains sulphuric acid and some greasy material from petroleum.

* Some Miscellaneous Wood Oils for Flotation, this volume, p. 646.

David Cole.—I believe that you are making sulphuric acid very cheaply at Anaconda. Since sludge acid consists of coal oil and sulphuric acid, I have wondered if you could not compound it at Anaconda more cheaply than you can buy it.

E. P. Mathewson.—We have made sludge acid at Anaconda, but it is more expensive than that on the market. We got good results with the acid we manufactured.

Rudolf Gahl, Miami, Ariz.—Some of the wood oils have the drawback that the concentrates produced with them have the tendency of retaining some of the gangue very firmly and it is difficult to settle such concentrates. We have found very few we could use. We have found some pine oils of the more refined character, but we cannot do anything at all with pine tar. Hardwood tar may have different qualities, and I would like to ask Mr. Ralston and Mr. Allen if they have given some attention to the physical characteristics of the froth made by the wood oils which they have tested.

O. C. Ralston.—Of course, the tests reported in our paper on wood oils are laboratory flotation tests, and give the flotative value of the oil and do not discuss later mechanical treatment of the concentrates. We cannot tell what will happen later in that part of the mill. That is another point. Sometimes you cannot tell even when you get to the mill what you can do with such a froth. I think Mr. Cole's patent for breaking froth is a rather interesting sidelight on that subject. To answer Dr. Gahl directly, our suggestions are only based on laboratory experiments on the flotation problem alone.

Rudolf Gahl.—Mr. Ralston has done so much work on the theoretical end of flotation, he might be able to answer a question which I have often put to myself. Why is it that at Anaconda they can float chalcocite ore (Is it not chalcocite ore?) with acid and when we add acid here we spoil our flotation altogether?

O. C. Ralston.—Some day when I understand what flotation is, I will answer that.

Norman Carmichael, Clifton, Ariz.—One point which has not been touched on in this discussion today: In going through the Old Dominion concentrator, I noticed that they were using what appeared to be caustic soda. I think if there is anyone here who can give us any explanation in regard to the use of the caustic soda, it would be interesting.

W. B. Cramer, Globe, Ariz.—The point which Mr. Carmichael has just spoken of, I shall try to answer. I was interested in what Mr. Mathewson said about the use of acids at Anaconda. I believe the analyses of the Anaconda and Old Dominion slimes would be much the

same. We find, as Dr. Gahl finds, that the moment we use acid in flotation, our flotation suffers considerably. In fact, it almost ceases. If we use a pulp slightly alkaline, the flotation work is satisfactory. Caustic soda is expensive, and we are using about 1 lb. to a ton. A distinct advantage that caustic soda has at Old Dominion—at present we are running on high flotation feed, averaging 3 per cent.—is in the matter of settling. We are installing a new Dorr thickener, and our capacity so far as thickening concentrates are concerned is very limited. Soda has the effect of flattening out the froth considerably, allowing us to cut down the water in the launders at machine less than half, which allows us to handle the concentrates in the Dorr thickener very well. Without the use of caustic soda, we may lose 4 or 5 tons of concentrates a day. It goes over and is recovered later in the secondary tanks. Why the soda should have this effect at the Old Dominion and the opposite at other places is hard to explain. We cannot use acid. We get better results with the caustic soda. In other places the opposite is found. At Nacozari a few years ago we were not able to get a satisfactory froth without acid. If we put acid in small amounts, the pulp still being non-acid, the froth was light in character; but increasing the amount of sulphuric acid until pulp was acid to methyl orange, a splendid froth was obtained.

C. W. Merrill, San Francisco, Cal.—But the use of caustic soda is solely for the purpose of decreasing the amount of water from the overflow?

W. B. Cramer.—Caustic soda does increase our extraction. So, in a matter of dollars and cents, it pays us to use caustic soda, but it brings the cost per ton for treatment up about 5 c. per ton. It does increase our extraction, increases the copper content of the froth and lowers the insoluble content.

C. W. Merrill.—I understand that the purpose is threefold; to cut down the water, increase the extraction, and clean the concentrates.

R. S. Handy, Kellogg, Idaho.—I would like to suggest that in the treatment of refining petroleum, acid is used, which before the treatment is finished is neutralized by caustic soda, so that the residue is sludge acid. If I am not mistaken, it contains a considerable quantity of caustic soda. That has been my experience in California. I would like to know if that is true.

O. C. Ralston.—The acid sludge from petroleum refining is usually a product obtained by the addition of strong sulphuric acid to the partially refined petroleum products in order to remove certain impurities, and is usually removed before sodium hydrate is added to the remaining oil to neutralize the small amounts of sulphuric acid held by the oil.

Hence it consists of strong sulphuric acid combined with some of the dark tarry constituents of the oil. Good flotation acid sludge comes from the California oil fields and titrates about 50 per cent. H_2SO_4. The other constituents are apparently the sulphonated asphaltic compounds contained in the oil. On that account sludge acid does not contain sodium hydroxide. Turning to the question of the use of sodium hydroxide in the flotation pulp and the reason for the good results obtained, I think that an explanation is easy. This substance, as well as other substances of alkaline reaction, like sodium silicate and sodium cyanide, is used in a number of mills which have a great amount of clay and other finely divided material in the ore pulp treated. Such an ore pulp is usually partially flocculated and sodium hydrate is known to be a powerful reagent for deflocculating certain gangue slimes. That is, its hydroxyl ions are adsorbed into the film of water in contact with the surfaces of individual particles, in greater proportion than are the sodium ions. This amounts to giving these particles a stronger electric charge of the negative sign and they fly apart (are deflocculated). In case any sulphide particles had been entrained in the flocs of gangue, this deflocculation liberates these sulphide particles for flotation. This explains the statements by Mr. Cramer that he gets better extraction and higher grade of froth by the use of caustic alkali in the pulp. I am informed by other operators that it also reduces their consumption of flotation oil as much as 50 per cent. in some cases. This seems reasonable, as the sulphide particles are all liberated and ready to be oiled so that oil will not have to penetrate to the center of a floc to oil the entrained particles of sulphides. Further, the great number of oils used are more easily emulsified in alkaline solutions. In the cases where the use of sodium cyanide has proven beneficial I am not sure that the alkaline reaction of the reagent is the sole contribution of this reagent to improvement of the flotation, but the case is interesting because Charles Butters and associates claim that it is an impossibility to float in the presence of cyanide. In at least two instances coming under my observation, the flotation of the sulphide minerals was improved by the addition of cyanide. It is just another instance of being told that certain things cannot be done in flotation, only to discover later that with proper care and right conditions they can be done. Don't believe any one who tells you that you can't float under some particular conditions. He may be partially right but the statement of negative results should always receive the qualifying phrase, "as far as I know, and to the best of my experience." For instance, Dr. Gahl claims that acid spoils flotation at Inspiration. I feel confident that the right conditions for an acid flotation of that ore could be found.

David Cole.—When we were at Chino a few days ago, we saw flotation machines handling vanner concentrates in which they were using

lkali and rosin as the flotation agent. Mr. Ralston looked at this, and would like to have him tell us what we saw.

O. C. RALSTON.—That solution is a solution of sodium resinate, the dea of the alkali being to get the rosin into solution. The rosin has a ery good effect on the froth, giving a particularly stable froth. They vere working under conditions where the oil froth wanted to die, and the ddition of the rosin was the proper thing to bring up its strength and llow it to rise until it got over the discharge board. I think there was 1o intended significance in the additional alkalinity. In that case the lkali was added for a particular purpose, simply as a solvent for the rosin.

THE CHAIRMAN.—I understand that Mr. Ralston wishes to get some uestions answered by the members of the Institute, and I will ask him o kindly present those questions.

O. C. RALSTON.—A great many of the questions I am interested in ave been asked and answered this afternoon, so that you will all welome the fact that the list can be cut down. This list was presented by D. A. Lyon, who sent it out in a circular letter all over the country. So far I have not heard of anyone who has prepared a list of answers or publication. I don't see why people need to be afraid to answer hem. The other day somebody said, "it is safe to talk about theories of flotation, because none of us know anything about them." I will only pick out a few questions. The first one is this: What is the effect of dilution of pulp with water on the flotation of the minerals contained? The reason for asking that question is, of course, obvious. There are enough large-scale operators who must have tested this question out. So, it will be interesting to get the consensus of opinion. I think that Dr. Gahl might help us.

RUDOLF GAHL.—Well, I would say offhand that the effect of dilution is to make a cleaner concentrate, and to make it more difficult to produce concentrate. I think it takes more oil to produce the same amount of concentrate from a dilute pulp than it takes to produce it from a thicker pulp.

O. C. RALSTON.—Might I supplement by asking you what determines the amount of oil necessary for flotation? Is it the amount of water you are using, making a certain strength of solution of frother in water, or is it the amount of mineral in the ore? If you had an ore consisting of 50 per cent. mineral, would you use more oil than if it contained 5 per cent.? What amount of oil must be used in flotation?

RUDOLF GAHL.—What amount of oil must be used in flotation? I do not feel qualified to answer Mr. Ralston.

O. C. RALSTON.—The Butte and Superior ore, containing 30 per cent. of mineral, as compared with the Inspiration ore with 5 per cent. or 10 per

cent. of mineral, nevertheless uses less oil, if anything. So, the proposition would be that the amount of mineral in the ore is not determinative of the amount of oil necessary. It must be the amount of water in the pulp which is determinative of the amount of oil necessary.

David Cole.—I think the kind of oil has a lot to do with the amount used.

O. C. Ralston.—That is a question not of theory but of experience. For instance, the amount of pine oils being used in the Coeur d'Alene district is often less than 1/5 lb. oil to the ton of ore, while at Anaconda they use several pounds of oil per ton. At Anaconda they use a considerably less expensive oil.

A. P. Watt, St. Francois, Mo.—Mr. Ralston has asked the question whether the amount of oil used in flotation is proportional to the water or to the solids in the feed to a flotation machine. My experience with the flotation of lead ores may be of interest. In one particular case a flotation machine was receiving a feed with a ratio of solids to liquid of 1 to 7. This ratio was later changed to 1 to 3.5 and the amount of oil used was decreased approximately 50 per cent. This particular case would seem to indicate that the amount of oil used in flotation is proportional to the water rather than to the solids in the feed. I also found that the extraction was increased by the use of a thicker pulp. This result would be expected, as the volume of pulp passing through the machine was decreased, thus permitting it to be acted upon for a longer time than would be the case with a thinner feed.

David Cole.—Speaking of the small amounts of oil, we carried out some experiments the other day at El Paso, and on a density of 7 to 1, approximately, on an ore that carried 6 per cent. lead, 9 per cent. zinc, and 1.3 per cent. copper, with the use of 2/10 lb. of cresylic acid per ton, we were able to take out a large part of the lead. Then, by the addition of 3/10 lb. of No. 350 pine oil per ton, we were able to take out nearly all of the zinc. The products made assayed as per the accompanying table.

	Per Cent of Weight	Ounces Au	Ounces Ag	Per Cent. Pb	Per Cent Cu	Per Cent. Fe	Per Cent. Zn
Lead concts	12.86	0.110	12.85	35.7	7.32	7.7	13.0
Zinc concts	25.08	0.025	2 67	4.4	1.30	3.9	23.6
Tailing	59.68	0 005	0.45	0.3	tr.	3.0	1.9

Note.—This was done with a single operation without the use of cleaner for the concentrates.

There was very little oil used—a total of 1/2 lb. per ton of ore. If we had used a larger amount of oil, we would not have been able to get that

separation. We put in just enough cresylic acid primarily to get that result.

C. E. Mills.—I think Mr. Gottsberger might contribute something. The fact is that the Miami Co. and the Inspiration Co. are treating similar ores, except that we have not as much copper as they have. We use quite a little more oil in flotation than Mr. Gottsberger does; and I think, perhaps, they have got more water in the pulp. Is that about the condition, Mr. Gottsberger?

B. B. Gottsberger, Miami, Ariz.—I am sorry to say I could not give the density of the pulp.

C. E. Mills.—We are using 1:3.

B. B. Gottsberger.—Our oil mixture is much thinner because we are using less coal tar. I think we have found that with a thick mixture composed very largely of coal tar it is really necessary to add the oil in the grinding mill. We do find, however, that it is not essential to add these oils in the grinding mill in order to get good flotation work. At present the necessary mixture is obtained by adding the oil in a bucket elevator.

C. E. Mills.—Can you give us another question, Mr. Ralston?

O. C. Ralston.—I might ask this question, which would supplement Mr. Handy's question: Are ores which contain very much fine colloidal material harder to treat successfully than granular ores? As far as I know, they usually are. Dr. Gahl has mentioned something about the treatment of them. I would like to pick on Dr. Gahl again to answer that.

Rudolf Gahl.—I would say that colloidal ores are harder to treat than granular ores.

O. C. Ralston.—There is a question in my mind. Is it possible to treat by any method of flotation colloidal material? Would it be possible to deflocculate that material and separate the granular material? In our experimental laboratory we did work on material sent us by Mr. Watt which approximated that condition. It was from Missouri, in the disseminated lead section, where the leady limestone had weathered away, leaving practically nothing but clay. The galena oxidized to lead carbonate, and Mr. Allen's attempts to sulphidize and float that ore were completely unsuccessful. It was practically nothing but a clay—about 100 per cent. colloidal material. So, a question arose: How much colloidal material is allowable in an ore?

C. W. Merrill.—Do you mean only colloidal gangue or the whole ore to be of colloidal nature?

O. C. Ralston.—This question is very well put. No one seems to have studied the question as to whether only the gangue in an ore is colloidal. "Colloids" are blamed for a lot of trouble and it occurred to me that whether the flotative minerals are reduced to colloidal size and condition, or not, colloidal gangue might cause trouble.

W. B. Cramer.—Our experience at the Old Dominion on primary slimes differs from that at other plants. Our mill feed contains about 15 per cent. minus 200-mesh material which may properly be called "primary slime." This slime was separated in the first compartment of our fine jigs, thickened in a Dorr thickener and treated separately from the secondary mill slimes by flotation. An extraction of 84 per cent. was obtained in the flotation treatment of this purely primary slime. Hence it may be stated that the primary slime at the concentrator of the Old Dominion Copper Mining and Smelting Co. is as readily adapted to flotation as the secondary slimes.

C. W. Merrill.— I can conceive of colloidal ore where the mineral values were extremely finely divided that could not be treated.

A. P. Watt.—Referring to the ore from Missouri of which Mr. Ralston has just spoken. I believe that Mr. Ralston and Mr. Allen were unable to make any extraction by flotation on the sample of slime I sent them. When, however, we took a more granular sample we were able to make some separation. We found that a good extraction could be made on the granular portion if the colloidal portion were first eliminated. The colloidal portion of the ore, however, appeared to prevent flotation.

The sample of slime submitted to Mr. Ralston was extremely fine. Although it contained 2 per cent. lead no signs of cerusite could be detected when the slime was treated on a vanning plaque. Whether or not the cerusite was in a "colloidal condition," I do not know.

F. S. Schimerka, Clifton, Ariz.—In connection with the question brought up a short while ago as to the different quantities needed of oil to successfully float, I wish to ask one question. Has it ever been noticed by the flotation experts present today whether it makes a difference as to what density the pulp is in when the oil is added in the grinder or mixing machine—whether a pound of oil goes farther at a high density?

The Flotation of Minerals

BY ROBERT J. ANDERSON,* B. S., ROLLA, MO.

(Arizona Meeting, September, 1916)

INTRODUCTION

DURING the past 5 years no subject has aroused more interest or received more attention among mill operators than flotation. One reason for this is, undoubtedly, the remarkable success of the process in Australia where that country has risen to the position of furnishing one-fifth of the world's supply of zinc. The plant of the Butte & Superior Copper Co., designed by James M. Hyde in 1912, was the first important large flotation plant in this country; other installations have been made since then with considerable frequency, particularly so in the last 2 years.

Without going into complete detail, it may be safely said that although flotation is a highly successful commercial process, it is more or less on an empirical basis. Upon scanning the literature, it is found that the flotation investigations have, in the main, dealt with a solution of the problems which accrue to practise; until recently no attempt was made to remove the difficulties in the way of the formulation of a consistent and harmonious theory. Of late this phase of the work has been receiving considerable attention, particularly by the United States Bureau of Mines at its Salt Lake City station, by the Mellon Institute at Pittsburgh, by the General Engineering Co. at Salt Lake City, by many of the mining schools, and private individuals almost *ad infinitum.* There is, at present, an enormous amount of laboratory work being carried on in flotation research, and probably the time is not far away when a generally acceptable explanation of flotation phenomena can be set forth. However, at this time, it seems as though the current technology is befogged with a superabundance of contradictory evidence; this confusion can only be dispelled as time proceeds and the knowledge of the subject becomes more complete.

Many phenomena are supposed to contribute to the flotation of minerals, whether in whole or in part is a mooted question. I shall only sketch roughly the present tendency of ideas and make no reference to the first early and crude notions which have little value other than histori-

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cal interest. Many questions have arisen in connection with flotation, concerning such matters as surface and interfacial tensions, adsorption, absorption or occlusion, colloids, emulsions, electrolytic and electrostatic phenomena, etc. These phenomena are to be discussed in later paragraphs in order to ascertain what bearing they may have, if any, on froth flotation. No final solution of this very large problem is attempted in this paper, for in any discussion of flotation theory, at this time, one is confronted by so many obstacles that at best only a rather fragmentary presentation can be made. An attempt has been made in this writing to correlate the operative and fundamental principles and compare in a measure some of the theories so far advanced.

Discussion of Certain Factors

Surface and Interfacial Tension

Surface tension has been rather well defined in articles appearing in the *Journal of the American Chemical Society* during the years from 1908 to 1913. The theory of surface tension has been treated in particular by Laplace, Gaus, and more recently by Van der Waals, and by Willows and Hatschek.[1] As defined by Jones,[2] "potential energy, present at the surface of liquids, produces a tension which is known as surface tension." The phenomena which are invariably indicative of surface tension are commonly observed: Drops of a liquid not exposed to an external force, *i.e.*, either suspended in another liquid of the same specific gravity or freely falling, assume a spherical shape, the sphere being that form of body with the smallest surface per given volume; further, if water be placed in an open vessel its surface film will be a measurable quantity, and its thickness will vary with a number of factors of which temperature is one. Its thickness is observed as ranging from 4×10^{-5} cm. to 4×10^{-8} cm., and its density, when referred to the main bulk of the water below, will approximate 2.14. Surface tension is not affected by the surface area. It is numerical in value and expressed in dynes per centimeter. It is a variable factor dependent on temperature, increasing numerically with falling temperature, *e.g.*, water at 18° C. has a surface tension of 73 dynes per centimeter, and at 0° C. 75 dynes per centimeter. At the critical temperature of a liquid its surface tension becomes nil.

All liquids have a definite cohesion or tensile strength, which is ascribed to the well-known mutual attraction of their molecules. This then is comparable to a pressure existing within a liquid, which has been termed the "intrinsic" pressure. Naturally the value of the surface tension of solids is numerically a high one. The surface tension of a pure liquid

[1] Willows and Hatschek: *Surface Tension and Surface Energy*, 1915.
[2] Jones: *Elements of Physical Chemistry*, 1907.

against its vapor can be and is markedly affected by the addition of soluble contaminants. Some salts will raise the surface tension of water while others will lower it; the fact that the salts of weak acids will lower the surface tension of water is explained by the fact that free acid is liberated by hydrolysis. It is further known that all acids will lower the surface tension of water. The surface tension of water is decreased by the addition of oil, or better, oil will reduce the interfacial tension between the water-air phases. A phenomenon for which no explanation has been given is the one which shows that the addition of contaminants may either raise or lower the surface tension of water, but such addition, while it may decrease that tension greatly, can increase it only slightly. Any lowering of surface tension is more marked in a liquid which has a high surface tension, *e.g.*, water, than in liquids of low surface tension.

There can be, of course, no surface tension without adsorption, which produces, in the case of positive adsorption, an increased surface concentration resulting from a lowering of the surface tension by the contaminating and dissolved substance, whatever it may be. The equation of Gibbs—$u = - c/Rt \cdot d\sigma/dc$—gives the relationship between surface tension and the distribution of the solute between the bulk of the liquid and the film interface. Here the notation is:

u = excess of substance in the surface layer,
c = concentration in the main body of the liquid,
R = the gas constant,
t = absolute temperature,
σ = surface tension.

This shows that when the surface tension is reduced by the addition of a contaminant, the quantity $d\sigma/dc$ is negative and u is positive (from algebraic consideration). The surface film then contains more of the contaminant than the main body of the solution. If the surface film contains less of the contaminant than the main body of the solution it is a case of negative adsorption.

As given in the foregoing, the surface of a liquid against its vapor is in tension—surface tension; the surface of liquid against another liquid, or a gas or solid, is also in a state of tension: this is termed interfacial tension. In the consideration of the bearing which surface and interfacial tensions have on froth flotation, this condition of affairs obtains in the flotation machine: Pulp consisting of ore of approximately 80-mesh, water in ratio of 3:1 of ore, and oil in disappearingly small amount, is being violently agitated. For the sake of a specific case, the air is being forced mechanically into the swirling pulp by beaters or stirrers. These phases then are present in flotation by the oil-froth process, viz., solid-liquid (ore-water), solid-liquid (ore-oil), solid-gas (ore-air), liquid-liquid (water-oil), liquid-gas (water-air), and liquid-gas (oil-air). Thus six ten-

sions are present, but if the oil is soluble in the water the tension number becomes three. It is known that pure water can not be made to maintain a persistent froth because its surface tension is too high. Acid, if present, will lower the surface tension of water, as will oil if it is soluble to any extent. Very many interesting and important, as well as speculative, consequences follow from a consideration of interfacial and surface tensions, and, in a later paragraph, we shall have occasion to return to the consideration of these complicated phenomena.

Certain metallic sulphides, *e.g.*, galena, have the power of floating on undisturbed water; they are not wetted and the curve of contact is convex. Some gangue minerals, *e.g.*, quartz, possess an adhesive force of attraction for water which exceeds the intrinsic pressure of the water; they are therefore wetted and sink to the bottom, being drawn through the surface film. Such properties of the minerals are affected by the presence of oil, acid, and other reagents. Oil has a greater adhesive attraction for sulphide minerals than for gangue minerals; and the addition of acid and oil (if it is soluble) acts as a contaminant which will lower the surface tension of the water and aid in the production of a persistent froth. Let us now look into the question of adsorption and see what part it plays in flotation, since it is so requisite to the production of a variable surface tension.

Adsorption

Generally speaking, adsorption deals with the unequal distribution of substances at the interface between dissimilar phases such as, solid-solid, solid-liquid, solid-gas, liquid-liquid, liquid-gas, and gas-gas. It is purely a physical effect. Commonly, adsorption[3] is construed to be the result of the condensation of a disperse phase upon the interfacial boundary solid-liquid. Returning for a moment to the Gibbs equation mentioned above, adsorption may occur if the interfacial tension solid-liquid is reduced—this being positive adsorption. If, however, such an interfacial tension is raised in value it is a case of negative adsorption as the solute or disperse phase will be rejected from the surface. Any condensation, strictly stated, of a solute or disperse phase in the interfacial boundary separating liquid-liquid or liquid-vapor is held to be a special case of adsorption. However, in the general sense, the phenomenon is looked upon as being the result of condensation of a disperse phase in the interface of two immiscible phases. Adsorption is shown very strikingly by colloid gels—the product obtained by the coagulation of sols—and certain cases of selective adsorption are very remarkable. Adsorption will naturally vary with the surface exposed. In Miss Benson's experiments with amyl alcohol in aqueous solution, amyl alcohol reduced the surface ten-

[3] Briggs: *Journal of Physical Chemistry*, vol. 19, No. 3, p. 210 (March, 1915).

sion of the water, and it was found by producing a voluminous froth that the alcoholic concentration in the froth exceeded that in the bulk of the aqueous solution by about 5 per cent. A froth has a very large surface, and it would be expected that the adsorption would be greater. Such experiments prove the value, qualitatively, of the Gibbs rule.

Recent work shows that all solids do condense certain amounts of gases on their surfaces and retain them there with very great tenacity. Liquids in like manner adsorb gases. Further, liquids and solids exhibit marked selective adsorption of gases. Although this selective adsorption obtains, no evidential proof has been submitted which says that the amount of gas adsorbed by one substance is largely different from the amount adsorbed by another substance. An electric charge on an adsorbed substance probably would considerably influence the amount adsorbed. The adsorption of air plays an important rôle in flotation, for as Breuer points out, the adsorbed air film is enormously responsible in preventing the coalescence of solid particles.

A comprehensive study of the adhesion of small particles of solid to the dineric interface (surface separating two liquid phases) has been made by Hofmann[4] based on the theory of Des Coudres.[5] From the standpoint of flotation this may be given as follows: If a solid particle, *e.g.*, quartz, is wetted much more strongly by water than by another liquid, *e.g.*, oil, the water will displace the oil, and a film of water will form about the quartz particle according to the relative forces of adhesion. Then the quartz particles will remain in the water phase if the water has a specific gravity greater than the oil, regardless of their size; but if now the oil has a greater specific gravity than the water, then the quartz particles will remain in the water phase until the size of the particles is such that the force of gravity will remove them from the water. Conversely, if a solid particle, *e.g.*, galena, is wetted more strongly by oil than by water, the oil will form a surface film about the particle and hence prohibit the particle from being wetted by water, *i.e.*, from entering the water phase. Then the galena will only enter the water phase when the water is more dense than the oil, and, further, when the galena particles are of such a size that the force of gravity overcomes the adhesion of the oil film to the oil.

Returning to purely theoretical considerations, Hofmann draws certain conclusions at this juncture which deal with the supposition that solid particles will then remain in the surface separating two immiscible liquids, if those particles are wet partially by each liquid. I quote Bancroft at length on this matter:[6] "The solid particles tend to go into the water phase if they adsorb water to the practical exclusion of the other

[4] *Zeit. Phys. Chem.*, vol. 83, p. 385, 1913.
[5] *Arch. Entwicklungsmechanik*, vol. 7, p. 325, 1898.
[6] Bancroft: *Journal of Physical Chemistry*, vol. 19, No. 4, p. 287 (April, 1915).

liquid; they tend to go into the other liquid phase if they tend to adsorb the other liquid to the practical exclusion of the water; while the particles tend to go into the dineric interface in case the adsorption of the two liquids is sufficiently intense to increase the miscibility of the two liquids very considerably at the surface between solid and liquid."

Any simultaneous adsorption of two immiscible liquids by a solid would probably tend to result in the formation of a homogeneous liquid phase at the surface of the solid.

In regard to the effect of contaminants or other impurities in contact with two immiscible liquids, this condition obtains: If the contaminant is soluble in one liquid but not in the other, and also lowers the interfacial tension of the two, the equation set forth by Gibbs exacts the requirement that the contaminant should obtain in the interface. Examples of this prove the validity of the law.

The terms adsorption and absorption have been used interchangeably in some writings, thus contributing to the already existing confusion of ideas.

Absorption or Occlusion

There are three ways by which gases can be held with reference to solids: viz.: (1) By surface adsorption; (2) in solid solution; and, (3) by occlusion. The term "occlusion" has been applied indiscriminately to any of the above methods by which gases are held by solids. Strictly speaking, by occluded gas is meant gas which is absorbed and held in finely divided pores or openings which may be of microscopic size. A recent theory[7] holds that occlusion plays the operative rôle in the flotation of minerals by all processes. I am unable to reconcile myself to this explanation for a number of reasons. Marked instances of occlusion at normal temperature are known only in certain amorphous substances like charcoal. Many metals, of course, both in the liquid and solid states, have the power of occluding gases, often in marked degree. There may be and undoubtedly are fine pores in the floatable minerals, which may in a sense be considered as an assemblage of capillary tubes; these can and do occlude gas. Yet occlusion is marked only in amorphous substances and in certain metals as just stated. It is definitely known that occluded gases are retained with very great tenacity by the substances occluding them and therefore expelled only with difficulty. It seems anomalous then to hold that the occluded gas can depart from the mineral occluding it with sufficient speed to aid the air bubbles in the liquid in the process of flotation. I believe rather firmly that occlusion is not a cogent factor in flotation by any process, and that a more con-

[7] Durell: *Mining and Scientific Press*, vol. 111, No. 12, p. 428 (Sept. 18, 1915) and Durell: *Metallurgical and Chemical Engineering*, vol. 14., No. 5, p. 251 (March 1, 1916).

sistent theory may be formulated without postulating these conjectures regarding occlusion.

Colloids

Colloids, in the original definition of the term by Thomas Graham, are not a definite class of substances; rather a large number of different substances may be made to assume the colloidal state if proper precautions are taken. All of which reveals the striking fact that this colloidal condition is a *state* and not a *form* of matter. The ultra-microscope of R. Zsigmondy and H. Siedentopf has increased the knowledge of colloids to a great extent. A rather general statement may be made regarding colloids and that is, that they do not show osmotic pressure in any appreciable amount. Colloidal solutions—sols—are regarded as systems of two phases, in which the dissolved substance is the disperse phase and the solvent the continuous phase.

Since in flotation the ore is often as small in size as certain of the colloids, the flotation pulp (ore, water, etc.) can be looked upon as a coarse suspension, and the laws of colloids apply here with equal force as in the realm of colloidal chemistry. So-called suspensions are systems consisting of solid particles of microscopic size distributed through a liquid. As mentioned by Ralston,[8] Reinders has treated at length the particular case of a solid phase maintained in contact with two liquid phases, *i.e.*, two immiscible liquids. His work is based on the different interfacial tensions existing, and his experiments and those of Hofmann, as mentioned in an earlier paragraph, have considerable bearing on the flotation problem.

Emulsions

Emulsions are fairly coarse dispersions of one liquid in another with which it is immiscible. The simplest and commonest emulsions are the oil-water emulsions, *i.e.*, the pure oil-water emulsions, containing no emulsifying agent such as soap, proteids, etc. In such systems the oil globules can be coagulated by electrolytes, show the Brownian movement strikingly, and can even be retained by some filtering media. Any process of emulsification is dependent on a surface tension lowering, or, to be more precise, on a lowering of the interfacial tension between the two phases. According to Briggs and Schmidt,[9] the two essential requirements of an emulsifying agent are these: (1) The property of condensing by adsorption in the dineric interface; and (2), the ability to form under these circumstances a strong coherent film. Temperature is a cogent factor in emulsification, for its effect is to reduce the interfacial

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[8] Ralston: *Mining and Scientific Press*, vol. 111, No. 17, p. 623 (Oct. 23, 1915).

[9] Briggs and Schmidt: *Journal of Physical Chemistry*, vol. 19, No. 6, p. 479 (June, 1915).

tension between phases and also to lower the viscosity of the phases. In the production of emulsions, a considerable amount of surface energy is produced because of the relatively large surface area of the disperse phase; an emulsion is the more speedily effected if such surface energy be reduced by the use of a liquid having a low surface tension as the continuous phase. Some emulsions, under certain conditions, display a great increase in viscosity over that of either of the immiscible phases, *e.g.*, the emulsions of the Pickering order—up to 99 per cent. of oil in 1 per cent. of soap solution—can be cut into cubes. Any emulsion produced with soap solution is at once rendered *nil* by the addition of acid, as the latter will decompose the soap.

If solid particles are suspended in a liquid, they tend to increase the viscosity of that liquid only gradually, depending on the amount of solid particles present. Experiments have shown that whenever a substance which is in suspension is wetted by two immiscible liquids simultaneously, it will go into the dineric interface in the manner already referred to, and will therefore tend to produce an emulsion. If, however, the suspended particles can not coalesce due to adsorbed oil film, or for other reasons, and thus effect the production of a coherent film, the emulsion will not be stable. Very few data are available on the production of emulsions by the oils used in flotation work, or on the matter of interfacial tensions between such oils and water. However, we are no doubt dealing with emulsified or partially emulsified pulp in some of the flotation processes, *e.g.*, the oil-froth process at least.

Electrolytic and Electrostatic Phenomena

Any substance which is placed in contact with water or many other liquids will assume an electric charge, the origin of which is, as yet, not set forth. Most substances when in contact with water become negatively charged, but these charges can be differed at will or reversed by the addition of the proper electrolyte in requisite amount. These electric charges are by no means confined to submicroscopic particles but are found also on the particles of a coarse suspension. Gangue minerals, *e.g.*, quartz, when suspended in water, are negatively charged, and sulphide minerals, *e.g.*, pyrite, are positively charged under like conditions. Oil drops are negatively charged, as are air bubbles under certain conditions which will be given subsequently. These charges are very minute when referred to the mass of the particle. Substantial evidence is at hand which goes to show that floatable minerals have the positive sign of electricity when suspended in water or can be made to assume that sign by the addition of proper electrolytes in sufficient amount. As Callow[10] observes, there is a parallelism between electrostatic character-

[10] J. M. Callow: *Bulletin* No. 108, p. 2334 (December, 1915).

istics and the flotation properties of ores. Many of the electrostatic principles have either been carried too far or misapplied as recent work shows.

Experiments in the realm of colloid chemistry indicate that the contact films are charged and that such charges markedly affect the dispersion or coherence of the particles in suspension. Naturally, oppositely charged contact films will coalesce while similarly charged contact films will repel each other, if the charges are sufficient in amount to overcome the force of cohesiveness; in the latter, dispersion is the result. The oil and air contact films having negative charges would tend to attract the sulphide particles, but further than this possibility electrostatics probably plays little part in flotation for the reasons given in the following paragraph.

As referred to above, it is pretty generally admitted that only minerals which are good conductors are suitable to flotation. Now then, as the electrical theory contends, electrified bubbles must be supplied to float the conducting minerals which are attracted, leaving behind the minerals which are not. The bubbles in flotation are simply air spaces contained by a mantle of oil or of water, and there is, therefore, nothing within to bear the charge. In case it could carry a charge, which would only be possible by the presence of contained ionized gases or water vapor, the charge would be speedily dissipated by contact with the interfacial boundary. Then in order for a bubble to carry a charge it must be protected by a dielectric film. Further, electrostatics plays probably little part in holding the sulphide particles and the gas bubbles together as neither the bubble nor the particle can have a charge of sufficient magnitude when referred to the size. The electrical theory has been strongly championed by at least one writer[11] and has been tolerated by some others. A recent article[12] indicates that the principles of electrostatics have been considerably misapplied. It is my belief that electrostatics may be a small contributing factor in flotation in a manner not as yet understood because of a lack of data concerning charges at the interfacial boundary between immiscible phases, *e.g.*, where the colloidal state is introduced in oil-water emulsions. Apparently, as far as any data at hand are concerned, they all serve to condemn any attachment of great importance to the electrical theory.

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[11] Bains: The Electrical Theory of Flotation, *Mining and Scientific Press*, vol. 111, No. 22, p. 824 (Nov. 27, 1915).

Bains: The Electrical Theory of Flotation, II, *ibid.*, vol. 111, No. 24, p. 883 (Dec. 11, 1915).

[12] Fahrenwald: The Electrostatics of Flotation, *ibid.*, vol. 111, No. 11, p. 375 (March 11, 1916).

The Character and Formation of Flotation Froth

Bearing in mind the phenomena discussed above, let us turn to the flotation of minerals by means of the froth process and look into the character and formation of froth. A froth has been defined as a multiplicity of bubbles; this seems to be wanting in some respects but will be requisite and sufficient for this purpose. The froth of flotation is formed by the action of air or gas in water containing impurities or contaminants, for pure water will not maintain a froth.

Froth and Bubbles

The idea has been abandoned of late, by most people, that a low surface tension is the essential requirement for froth formation. As mentioned by Coghill in a recent writing, the contamination of the liquid with an impurity which will cause a variable surface tension is the real requirement. A bubble of air is spherical in shape and this shape can only be maintained if the external pressure exceeds the internal pressure. Since a bubble does not expand *per se*, large bubbles can only be accounted for by heat, coalescence or electrification. Viscosity is an important factor in froth persistence as it increases the tenacity of the liquid film and thus prevents ready rupture. The rupture or bursting of bubbles is accounted for by these reasons:

1. Concussion upon a surface film which is deficient in the requisite viscosity and variable surface tension.
2. Relief of pressure—here the gas of the bubble in expanding exerts a pressure which exceeds that of the liquid film.
3. Adhesive force of the entrained gas for the atmospheric air.
4. Evaporation of the liquid film.

Flotation bubbles will burst then for any one or a combination of all these reasons.

Solutions in which the continuous phase is a solution of soap, various products from the saponification of albumens, etc., will froth voluminously even in a very diluted condition; frothing never occurs in pure liquids and is a definite proof that the solute or disperse phase lowers the surface tension of the solvent. A froth, which shows adsorption at the interfacial boundary of solution and gas, depends for its persistence on the production of a viscous film at that boundary; these viscous films are the direct result of surface adsorption of the disperse phase, *i.e.*, dissolved contaminants, the amount of which is small—disappearingly so. The work of Hall and of Miss Benson shows that in a foaming liquid the foam is richer in dissolved contaminant than is the bulk of the liquid. Froth formation in the Callow cell is the result of the injection of air into the pulp (already emulsified); the froth continues to obtain as long as

there is sufficient air injected into pulp of the proper consistency. The nature of the froth in the Callow cell is governed by the kind of oil used and by the amount of air. A pneumatic froth is unstable or ephemeral, *i.e.*, it dies rapidly when removed from the influence of the injected air. The mechanical froth, on the other hand, is thick and persistent, and must be broken up in dewatering the concentrates.

Oils

Oils have a well-marked selective action for metallic sulphides, tellurides and some other minerals. The fact that both the oil and the air or other gas have a selective adhesion for sulphides prevents them from being wetted by water. Conversely, the quartz and other minerals exhibit just the opposite characteristics. The gangue minerals, generally, do not exhibit adhesion for either gas or oil and hence they are readily wetted by water. Gases have a well-defined adhesiveness for oils; hence the air or gas adheres strongly to the oil film. The stability of a froth depends, in the main, on the kind of oil used, *e.g.*, pine oil makes a weak brittle froth, and creosote makes a stable elastic froth. The work of Devaux[13] on oil films explains how so small an amount of oil as is used in the various flotation processes can be so efficacious. From a consideration of the immiscible oil-water interface, if any oil will film the internal surface of a gas bubble the sulphide mineral particles would be contained in the oil-water interface no matter what the nature of the gas contained by the water film. The sulphide if it enters the oil phase would then present an oiled surface to the water phase. There are three conditions then: (1) The mineral enters the oil phase completely; or (2) the mineral enters the water phase completely; or (3) the mineral enters the oil-water interface.

Some experiments made to determine the nature of the frothing power, selective and collective action of different oils show some interesting results. I made some tests on a zinciferous slime from the Joplin area with different oils and the results obtained indicate that a definite mixture of oils will effect better recoveries than any one oil alone. The best combination consisted of pine oil as a frother, plus wood creosote as a frother and selector, plus refined tar oil as a froth stiffener. The complete results of these experiments will be given in a later paper.

In general, pine oil makes a brittle froth which immediately dies; creosotes make a more elastic froth, the bubbles of which may expand to 3 in. in diameter or more before rupture. Coal-tar products are rather poor frothing agents and if used must be aided by either creosote or pine oil to produce a good froth. Oils of a lubricating nature seem to be of

[13] Devaux: Oil Films on Water and on Mercury, *Smithsonian Report of* 1913, p. 261.

little value as flotation oils, while such light oils as gasoline and naphtha are of value only for thinning the heavy coal and wood tars.

Air and Gas

At this time, there are three ways by which a gas may be forced into a solution mechanically, viz.:

1. By beating it into the solution by means of beaters or paddles—as in the Minerals Separation and similarly mechanically agitated machines.

2. By pneumatic means—as in the Callow cell where the air is divided by the porous blanket bottom into minute sprays.

3. By so-called liquid jets—as in a process recently patented in which the air is introduced as a surface film surrounding a liquid jet by surface tension.

Conversely, there are three methods by which dissolved gas may be expelled from a liquid, viz.:

1. When the liquid is supersaturated, the excess gas is expelled.

2. By heating the liquid, when some of the gas is expelled due to an increase in its volume.

3. By pressure reduction, as in the Elmore vacuum process, where, according to the law of Henry,[14] "the amount of gas dissolved by a liquid is proportional to the pressure to which the gas is subjected."

An air or gas bubble on being introduced into a liquid is at once surrounded by a film of the liquid. Such a bubble will rise to the surface (carrying the metallic sulphides by reason of the forces already mentioned) on account of gravitation, by which is understood that the adherence of the air to the liquid is less than the force of gravity.

Résumé

From a consideration of the foregoing, it is believed that the theory based on the different interfacial tensions involved is the dominating one as well as the more consistent at this time. Probably flotation is due to a combination of phenomena which are rather high in the scale of complexity. The theory based solely on occlusion goes "by the board" as it has been shown that the contributing effect of this phenomenon has been interpreted rather laxly.[15] The phenomenon of electrostatics may be a small contributing factor but recent work indicates that the principles have been misapplied. An explanation more in consonance with

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[14] Jones: *Elements of Physical Chemistry*, p. 177, 1907.

[15] Ralston: Why Do Minerals Float? *Mining and Scientific Press*, vol. cxi, No. 17, p. 623 (Oct. 23, 1915).

fact can be given in terms of the interfacial tensions involved, without postulating either occlusion or electrostatics.

The main and essential requirements for froth flotation are: (1) The production of a persistent froth by any means; (2) the attachment of the bubbles of air to the sulphides or other material to be floated; and (3) the maintaining of a selective action of the froth bubbles for the sulphides or other material to be floated.

Why this is accomplished is outlined in the above, particularly under the subhead, "Adsorption." Probably, before any generally satisfactory estimate of the many complex factors, which are supposed to be involved, can be secured, a more thorough investigation of all of them will have to be made than has thus far been attempted. However, in this line of inquiry there has been a steady progression of thought and a remarkable increase in knowledge in the past few years; and the art of flotation will continue to improve and develop as knowledge of the different factors involved increases and allows the exercise of a better control of them.

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The style is unhappily difficult, so that the following suggestions may not be amiss. After several readings of the short paper on The Theory of Colloid Chemistry, read first the excellent summaries at the ends of the Theory of Emulsification papers before undertaking a rapid survey of the whole set; follow this by a more careful consideration of the summaries with re-reading of portions of the text when necessary; continue this process until the drift of the argument begins to reveal itself.

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DISCUSSION

OLIVER C. RALSTON, Salt Lake City, Utah (communication to the Secretary*).—The literature on the theory of flotation has been enriched, of late, by the views of a number of excellent mining engineers who unfortunately were tyros in physical chemistry and physics. Hence the obscurity and mystery with which the process is supposed to be surrounded.

The present paper goes far toward explaining some of the ideas which have seemed obscure or which have been poorly expounded by the men who advanced them, but there are a number of places where the author's explanations are unsatisfactory, or where he has fallen into the same mistakes made by the original propounders of the theories involved.

While recognizing the value of constructive compilation, it is nevertheless disappointing to find a paper reviewing the recent theories of flotation which presents nothing new—not even a new viewpoint.

At the bottom of page 529, there are stated to be present the following "phases" in flotation: "solid-liquid (ore-water), solid-liquid (ore-oil), solid-gas (ore-air)," etc. The writer could not possibly have

* Received September, 1916.

meant to apply the word "phases" to the ideas involved. Each of the *pairs of phases* mentioned has an interface in which exists the interfacial tension that he is talking about.

On page 535, in discussing electrified bubbles, it is stated that "the bubbles in flotation are simply air spaces contained by a mantle of oil or of water and there is, therefore, nothing within to bear the charge. In case it could carry a charge, which would only be possible by the presence of contained ionized gases or water vapor, the charge would be speedily dissipated by contact with the interfacial boundary. Then in order for a bubble to carry a charge it must be protected by a dielectric film." It can be seen from this series of arguments that the author has fallen into the same error that has confronted a number of exponents of an "electrostatic theory" of flotation. The theory of Bains,[1] which is objected to by Fahrenwald,[2] holds that the friction of the liquid and solid and gaseous masses on each other generates static electric charges and that the function of the oils used in flotation is to insulate these charged particles. Fahrenwald's objections to this theory are sound. However, all these writers have failed to consider the mechanism of attachment of electric charges to colloidal particles. They apparently have not understood how suspended particles, including air bubbles, can carry electric charges when suspended in a conductive solution. I have already called attention to the fact[3] that the *film* of liquid in contact with some other phase (gas, liquid or solid) has different properties from those of *bulk* water (or liquid). Mr. Anderson has been kind enough to quote some of the properties of this film, such as its thickness, density and tension, but has failed to remember that I also gave the difference in potential between its outer face and the bulk liquid, even when that liquid is highly conductive. Powis[4] has very recently expanded on the conception of charged colloidal particles and the theories of colloid chemists as to how such particles can hold electrical charges. Briefly, the charges are due to absorbed ions in the interfacial film. Any electrolyte in the solution is more or less ionized, and in case one of the ions is absorbed in the film more than the other, an apparent electrostatic charge is the result. In fact, the electric charge could not exist if the solution in question were not ionized, and hence conductive. It is a strange fact that the conductivity of the solution places serious obstacles in the way of the theory of Bains while conductivity (or ionization) is necessary to explain electric charges on

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[1] T. M. Bains, Jr.: Electrical Theory of Flotation, *Mining and Scientific Press*, vol. 111, pp. 824 and 883 (Nov.-Dec., 1915).

[2] F. A. Fahrenwald: Electro-statics of Flotation, *Op. cit.*, vol. 112, p. 375 (March, 1916).

[3] O. C. Ralston: Why Do Minerals Float? *Op. cit.*, vol. 111, p. 623 (October, 1915).

[4] F. Powis: Transference of Electricity by Colloidal Particles, *Transactions of the Faraday Society*, vol. 11, p. 160 (April, 1916).

the particles as viewed from the colloid-chemical or "capillary-chemical" standpoint.

On account of such serious misunderstandings as those above mentioned, it would seem that there is still room for the writing of a rigid critique on the theory of flotation.

As Mr. Anderson says, "in the line of development of flotation theory there has been steady progression of thought and a remarkable increase of knowledge in the past few years." That is just the trouble—the progression has been mainly in thought and too little in laboratory investigation. Too many have been willing to contribute *thoughts* to the technical press and too few have busied themselves with experimental measurements to prove the theories proposed.

An Explanation of the Flotation Process

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(Arizona Meeting, September, 1916)

Introduction

The flotation process for the concentration of ores is a method by means of which one or more of the minerals in the ore (usually the valuable ones) are picked up by means of a liquid film and floated at the surface of a mass of fluid pulp. Here they are separated from the other minerals, which remain immersed in the body of the pulp. In general, the minerals which are floated are sulphides of metallic luster, but some other minerals of metallic luster such as graphite and some sulphides with adamantine luster, such as sphalerite and cinnabar, are amenable to treatment by the process.

The importance of flotation lies in the fact that it is primarily a "slimes process" by means of which the particles of valuable mineral, too fine for efficient gravity concentration, are saved with a high percentage of recovery. Recoveries in the mills treating low-grade copper sulphide ores have been advanced 10 to 20 per cent. by the installation of the process and similar increased savings have been accomplished by the same means in mills treating sulphide ores of zinc and lead.

When finely ground ore containing sulphides mixed with a siliceous or earthy gangue is brought gently onto the surface of a body of water, in a direction forming an acute angle with the surface of the water, a considerable portion of the sulphide constituent of the ore floats on the surface of the liquid, while the gangue sinks. This is the so-called film flotation, exemplified by the Wood and Macquisten processes.

When gas bubbles are introduced into a fluid pulp composed of finely ground ore and water, to which has been added (1) a small amount of certain oils, or (2) a small amount of certain acids or acid salts, or (3) a small amount of certain alkalis or alkaline salts, or (4) a small amount of a mixture of oil with acid or alkali, the sulphide particles in the ore are brought to the surface on the gas bubbles. These collect in a froth

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heavily laden with sulphide particles. The gangue particles are not brought up by the bubbles, but remain in the mass of the pulp. This is the so-called froth flotation. Its variations, acid-froth, agitation-froth, and pneumatic-froth processes are discussed in detail later.

The points to be explained in the operation of these processes are: (1) The flotation of solid particles in a liquid the specific gravity of which is less than that of the solid; (2) the preferential flotation of the sulphide portion of the mass; (3) the functions of the reagents used.

Theory

The theory here presented in explanation of the points listed in the preceding paragraph appeals to the following physical phenomena. Surface tension, adsorption, adhesion and viscosity. The first three of these are closely related.

Surface Tension

Every liquid surface in contact with a gas or its vapor, behaves as if it were under tension. The value of this contractile force per unit width can be measured. Its value for water, 74 dynes per centimeter, is higher than for any other well-known liquid. (Liquid metals and fused salts are of course excepted.) This fictive tension is a convenient conception for many discussions and may be explained in terms of the intermolecular attractions of the substances forming the boundary. Another and very useful way of considering the phenomenon is to regard each unit of surface as having associated with it an amount of potential energy which is numerically equal to the surface tension. This relation is established as follows:

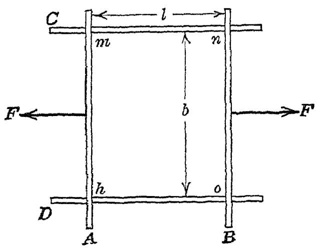

Fig. 1.

If two wires, A and B (Fig. 1), are so placed as to slide on two fixed wires, C and D, and if the wires, A and B, having been in contact, are separated a distance l against the pull of one of the surfaces only of a film *mnoh*, by applying the force F, the work per unit area will be

$$\frac{W}{A} = \frac{Fl}{lb} = \frac{F}{b} = T$$

which is obviously equal, numerically, to the force per unit width or the surface tension, T, of the one surface considered. Applying this conception of surface energy to different cases of contact, we can develop some statements of the relations of forces which are important in explaining many of the phenomena observed in flotation.

Consider first that two blocks of liquid, which have become rigid without any change in their other properties, are drawn together by the mutual action of their forces of molecular attraction until they perfectly unite over an area A. They would do an amount of work that we will represent by the letter W. Now consider that the same two bodies when brought near together, but not into contact, become liquid and unite over the area A. The work done is again W, but in this case it can be considered as due to the shrinkage of the surface by an amount $2A$, whence $W = 2AT = A(T + T)$, where T is the surface tension of the liquid. If two different liquids whose surface tensions are T_1 and T_2 respectively were brought together, the work due to the molecular attractions would be $(T_1 + T_2)A = W$, provided there was complete union, but if there is not complete union, there will be an interfacial tension T_{12}, and the energy equation becomes

$$(T_1 + T_2)A - T_{12}A = W$$

or

$$T_{12} = T_1 + T_2 - \frac{W}{A}$$

i.e., the interfacial tension T_{12} is the excess of the sum of the two tensions over the work which is done by them in allowing a unit area of the two liquids to come into contact. If a liquid is brought into contact with a solid, the energy equation is $T_{LS} = T_L - W_1$ (gas or vapor effects being excluded),

where T_L = the surface tension of the liquid,
W_1 = the work done in bringing a unit area of the liquid and solid into contact,
and T_{LS} = the interfacial tension.

If, therefore, $W_1 = T_L$, the solid has the same attraction for the molecules of the liquid as the molecules of the liquid have for each other and there will be no interfacial tension. If $T_L > W_1$ the interfacial tension T_{LS} will be positive; if $T_L < W_1$ there will be negative interfacial tension or a surface pressure. In the latter case the liquid will tend to spread over the solid.

Angle of Contact

When, as is the common case in the flotation process, there are three substances in contact, a system of forces as shown in Fig. 2 is brought into play. If O does not move indefinitely to the right or to the left, equilibrium will be attained when

$$T_{SL} = T_{GS} + T_{GL} \cos \theta$$

or

$$\cos \theta = \frac{T_{SL} - T_{GS}}{T_{GL}}$$

where T_{GS}, T_{GL}, and T_{SL} are the interfacial tensions or pressures at the gas-solid, gas-liquid, and solid-liquid contacts respectively. From this equation is deduced the important conclusion that as T_{SL} increases with respect to T_{GS}, the angle of contact θ becomes smaller (the gas and liquid being the same), or, in other words, the angle of contact is a measure of the tendency of one fluid to replace another on the surface of a solid.

We have examined the angles of contact of the water-air, oil-air, and oil-water surfaces against a number of the common minerals. We have found, in general, that the air-water contact angle is less for gangue minerals than for sulphide minerals; that the air-oil contact angle is less for sulphides than for gangues and less for any given sulphide than the air-water contact angle; and that the water-oil contact with solids takes the form shown in Fig. 3.

We found further that the invariable effect of oiling a solid surface is to reduce the air-water contact angle. This latter phenomenon is un-

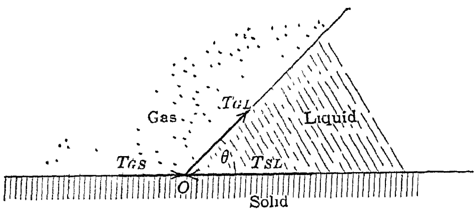

FIG. 2.

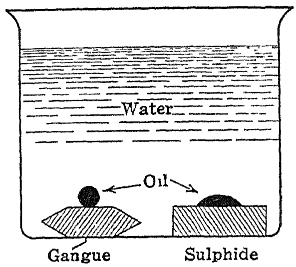

FIG 3

doubtedly aided by the reduction of the surface tension of the water due to contamination by the oil. This is further discussed later.

The conclusions forced by observation of the above phenomena are:

1. That water has a smaller tendency to displace air on the surface of sulphide minerals than on the surface of gangue minerals.

2. That the tendency of oil to displace air is greater at the surface of sulphide minerals than at the surface of gangue minerals.

3. That oil tends to displace water on the surface of sulphides and that water tends to displace oil at the surface of gangue minerals.

4. That water displaces air more readily on an oiled solid surface than on a clean surface of the same solid.

5. That these tendencies toward displacement are due to the interfacial tensions or pressures existing between the various substances, and that the resulting action of these interfacial forces is a manifestation of the tendency toward reduction of the total potential energy of the system. Wherever an increase in the solid-fluid interface will decrease the potential energy, such a change will occur.

These conclusions suggested the following confirmatory experiment.

A ring, 6.17 cm. outside diameter and specific gravity of 1.38, made of aluminum tubing, 0.63 cm. diameter, was cleaned and floated without trouble on the surface of pure water. The shape of the water surface at the air-water contact is shown in Fig. 4. The ring was then oiled slightly. The air-water contact angle was reduced, as shown in Fig. 5, to such an extent that it was impossible to float the ring. The same was true, as might be expected, when a cylinder of aluminum replaced the ring. A similar cylinder of glass tubing exhibited such a small air-water contact angle that it could not be floated.

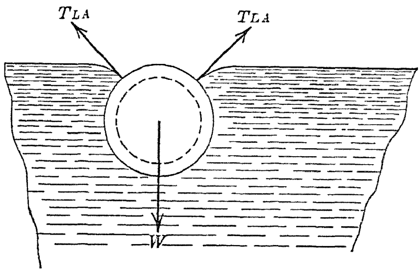

FIG. 4.

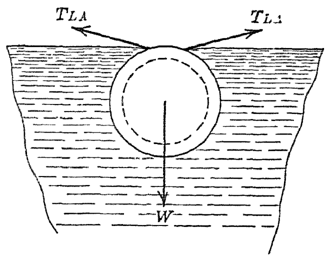

FIG. 5.

Adsorption

The surface layer between two physical phases is the seat of conditions of density and viscosity, also of apparent forces or energy manifestations, which are notably different from those in the bulk of either phase. On philosophical grounds it is impossible to consider that a real physical discontinuity occurs at the boundary between two media. In other words, there must be a very thin layer of transition in which there is a rapid but continuous change in the concentration of the components. This change in the concentration of a component at the interface is called adsorption, and may occur even between two phases which are ordinarily regarded as immiscible.

Adsorption at a gas-liquid interface may be demonstrated as follows: If a solid, which has been heated in a vacuum, is introduced into a measured volume of a gas over mercury in a calibrated tube, an amount of the gas will be adsorbed, as is shown by the change in pressure and volume compared to the space originally occupied.[1] These additional facts are established:

(*a*) The amount of the gas adsorbed at constant temperature increases with the pressure.

(*b*) It is different for different gases.

(*c*) It is different for different solids.

(*d*) It increases as the temperature decreases.

[1] Freundlich: *Kapillar Chemie*, p. 92.

(*e*) There is an energy transformation which is indicated by the heat developed through adsorption, analogous to the Pouillet phenomena mentioned later.

(*f*) Chemical reactions are assisted by the adsorbed layer.

It follows that the gas layers must vary in density, falling off rapidly with increasing distance from the solid. Quincke assumes that the density of the gas next to the solid is equal to that of the solid and concludes that the amount adsorbed will increase with the density of the solid. From these facts we conclude:

1. That gases and solids exhibit selective adhesion and that, therefore, gas bubbles will attach themselves more persistently to some substances than to others.

2. That this selective adhesion is a manifestation of a definite amount of energy possessed by each unit area of a gas-solid contact, and that this potential energy is capable of variation.

3. That chemical reactions which diminish this potential energy are aided by adsorption.

Adsorption at a liquid-solid surface manifests itself in a vacuum, or where the vapor phase is negligible, by the way in which the liquid spreads or gathers itself together on the solid; in other words, in the way in which the liquid wets or adheres to the solid. It is further manifested by an evolution of heat, known as the Pouillet phenomenon. A calculation of the condensation necessary to evolve this amount of heat, in the case of water against glass, indicates that the specific gravity of water in the adsorbed layer is increased to about 2.1.[2]

Adsorption of the gas at a gas-liquid surface is indicated

1. By the effect on the surface tension. The surface tension of a freshly formed mercury surface does not change in a vacuum, but falls off in the presence of different gases for about an hour. Certainly the density of a liquid cannot be constant at the boundary but must go over continuously into that of the gas.

2. By the increase in the solvent power of the surface.[3]

3. In the case of contaminated liquids, by the concentration of one or more of the components of the liquid at the gas-liquid surface. Every unit area of such a boundary possesses a definite potential energy which always tends to a minimum. If, therefore, the surface tension of a solution depends upon the presence of any component, such a change of concentration of that component will occur as will reduce the potential energy, *i.e.*, the interfacial tension. In other words, any component which reduces surface tension will be found in excess at the surface of a solution. For example, the surface tension of water is greater than that of alcohol. Experimentally, a drop of alcohol on a thin film of water

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[2] Lewis: *Phil. Mag.*, vol. 20, p. 502 (1910).

[3] Pockels: *Nature*, Mar. 12, 1891, p. 439.

rapidly reduces the surface tension of the water and the latter draws away from the alcohol. On the contrary, a drop of water on a thin film of alcohol spread over glass does not at once diffuse into the film but remains gathered in a heap. Such diffusion would increase the surface energy of the system, hence the water concentrates away from the gas-liquid surface. The greater viscosity of the surface of a solution above that of the bulk, or of that of a pure liquid, has long been recognized[4] by its damping effect upon a swinging magnetic needle and may properly be ascribed to gas-liquid adsorption. Closely connected with this is the formation of elastic solid skins or very viscous layers at a free surface, as, for example, in the case of solutions of peptone and dye stuffs. A peculiarity of saponine solution is the rigidity of its surface while the interior remains more mobile.[5] The surface of a freshly formed fairly concentrated solution of fuchsin is quite mobile, but in the course of a few

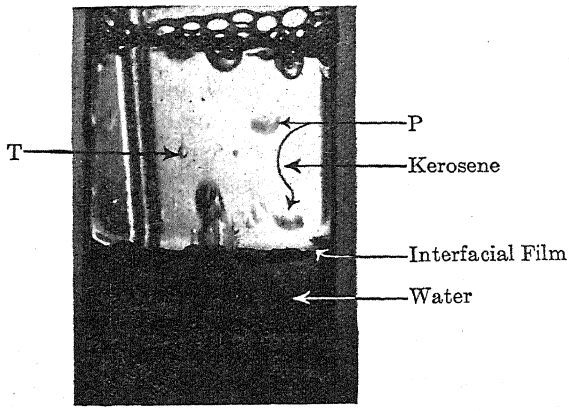


FIG. 6.

hours it changes to a reflecting skin with solid properties. Similar results are obtained with methyl-violet and peptone. In the case of the latter substance the skin is highly elastic. There seems to be no doubt that true adsorption is present here. In the case of crystal-violet, which closely resembles methyl-violet in its properties, a solution of 1 g. to the liter lowers the surface tension from 75 to 69.9 dynes per centimeter. Other causes for the production of a solid layer may, however, be present, for many of these substances in concentrated solutions stiffen into gelatines and since the concentration of the contaminant is great at the surface and the solubility has also a different value, the solid remains persistently. Either a chemically irreversible change or a transformation into a more difficultly soluble phase at the surface is clearly the explana-

[4] Daniells: *Physics*, p. 258, Fr. p. 76.
[5] Boys: *Soap Bubbles*, p. 115.

tion of the persistence of the froth in albumen solutions and the like. The properties of such surfaces apparently pass over imperceptibly into those of colloids.

Adsorption at the boundary between two liquids is evidenced by the effect on the interfacial tension just as in the gas-liquid solution surface, although the process is not one usually described as ordinary solution and one may also have to reckon with chemical reactions in the transition layer. With liquid-liquid surfaces, as in gas-liquid surfaces, the adsorption frequently gives rise to very viscous layers. The presence of such a viscous layer at an oil-water interface is easily shown by pouring any clear oil, kerosene, liquid vaseline, etc., onto water and then bubbling a gas through the water. Such an experiment is shown in Fig. 6. The interface has all the appearance of an elastic skin. Bubbles rising through the water and striking the under side of the interface stretch the film (see H, Fig. 7), and rising farther drag away a mass of water surrounded by this viscous layer. The system now appears as shown at M and rises to the oil-air surface on account of its lower specific gravity. Here the film, together with the excess water carried up as shown at C, breaks away and falls back through the oil, not in spherical form, as would be the case were the water drop not surrounded by a viscous film, but in hemispherical form (see P, Figs. 6 and 7) often trailing behind it a film with ragged edges, as it broke from the bubble. The tadpole-shaped water drops, T, (Figs. 6 and 7) are further evidence of the high viscosity of the oil-water interfacial film.

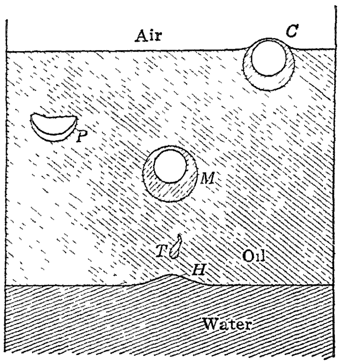


FIG. 7.

Viscosity

A marked increase in the viscosity of interfacial films is produced by the presence of finely divided solid matter. This increase is apparent in the experiment just described when finely powdered sulphide is thrown into the oil and allowed to settle to the interface, where it becomes entangled in the film. When gas bubbles are introduced, as before, the return water drops, coated with a film containing the solid particles, are much more irregular in shape than previously, and their coalescence after reaching the interface requires days or weeks. An even more convincing proof of the increase in viscosity of an interfacial film is given by the following experiment. If a needle is floated at the center of a surface of pure water in a beaker 4 in. in diameter and a chip of wood is floated

near the wall of the beaker, the needle may be caused to revolve by means of a magnet without disturbing the chip. If the surface of the water is dusted over with fine ore, the whole surface together with the chip moves as though it were a rigid solid.

Summary

The potential energy at a gas-sulphide contact is less than at a gas-gangue contact; hence gas bubbles will cling with greater persistence to sulphides than to gangues.

Oil replaces water at the surface of sulphide minerals.

Water replaces oil at the surface of gangue minerals.

Water replaces gas more readily at an oiled surface of a solid than at a clean surface.

The addition of any contaminant to water lowers the surface tension.

In any body of contaminated water there will be a concentration of the contaminant at the air-liquid surface.

Adsorption at a gas-liquid surface lowers the surface tension and increases the viscosity.

Adsorption at a liquid-liquid surface produces a film whose viscosity is higher than that of the bulk of either liquid.

The presence of finely divided solid matter in a film markedly increases the viscosity of the film.

Application to Commercial Flotation Processes

Film Flotation

Pulp with or without oil or acid is fed gently, at an acute angle, onto the surface of a body of still water. The sulphide floats and the gangue sinks.

Possible cases:

Case 1.—Sulphide, gangue, water.

Case 2.—Sulphide, gangue, water, oil.

Case 3.—Sulphide, gangue, water, acid.

Case 4.—Sulphide, gangue, water, acid and oil.

Case 1. *Sulphide, Gangue, Water.*—The governing factor in the initial flotation of the sulphide and immersion of the gangue is the difference in the air-water contact angle with the sulphide and gangue surfaces respectively. If the difference is great, as in the case with galena and quartz, good separation is obtained. After a considerable amount of sulphide has been floated, if the surface flow is sufficiently impeded, the particles congregate into clumps by the well-known phenomenon of apparent attraction of floating particles,[6] a scum is formed (whose viscosity and

[6] Hastings and Beach: *General Physics*, p. 156.

resistance to rupture are many times greater than that of the original water surface) and the floated particles are rendered more immune to immersion.

Case 2. *Sulphide, Gangue, Water and Oil.*—When oil is added in this process the phenomena are entirely different from the simple film flotation of Case 1. The oil concentrates at the surface since such concentration reduces the surface energy of the system. This adsorption of the oil at the gas-water surfaces causes the formation of a viscous film. When the mixture of sulphide and gangue is introduced at the surface, the sulphide particles tend to migrate into this layer and the gangue particles migrate into the water, for a sulphide particle in contact with oil represents the condition of least potential energy which is possible for the sulphide particle to assume in the system of oil, sulphide, and water. Likewise the gangue particle surrounded wholly by water represents the condition of least potential energy for the gangue particle to assume in this system. In this case also, the formation of a scum of floated sulphide increases the stability of the float.

Case 3. *Sulphide, Gangue, Water and Acid.*—The effects of acid are: (*a*) to diminish the surface tension of the liquid, (*b*) to diminish the gas-liquid contact angle, and (*c*) to increase the viscosity of the gas-liquid surface. The diminutions of (*a*) and (*b*) are more marked with gangue minerals than with sulphides. The result is that cleaner concentrate may be expected than in either of the previous cases, but at the expense of richer tailing.

Case 4.—*Sulphide, Gangue, Water, Oil and Acid.*—In this case a combination of results such as can be predicted from the preceding cases is obtained.

Froth Flotation

In order to explain froth flotation it is necessary and sufficient that the gas of each bubble shall be inclosed by a film of contaminated water which shall possess the following characteristics:

1. Low surface tension.
2. High viscosity.
3. A variable concentration of the contaminant (reagent).
4. A preferential adhesion of the bubble film to the sulphide mineral compared to that for the gangue minerals.

We will examine first the conditions required for the formation of a froth, or the continued existence of a thin film. Solutions which form froth are preëminently aqueous solutions and the properties of the liquid film are only secondarily determined by those of the gas.

The durability of a liquid film depends upon one or more of the following conditions:

1. A low surface tension which is locally variable so as to produce stable equilibrium.

2. High viscosity, which may pass over into
3. Chemical irreversibility and the production of solid skins.

Pure liquids do not foam; for example, water or alcohol. The reason is obvious. As the film is thinned out by stretching or by draining away of the liquid, the surface tension is reduced at some part below the general constant value. As soon as this begins the thicker and more powerful parts of the film drag away from the weakened parts which at once completes the rupture. These inequalities evidently would not be so marked or rapid in their operation if the surface tension were low. Increase of viscosity would also slow up the process. High viscosity and low surface tension do not occur in pure liquids. But the most important condition for durability is some means by which the equilibrium of the forces at any point in a film may be restored, when a variation of some of the forces occurs. In the case of a compound liquid or solution this is effected by the adsorption or change of concentration of one or more of the components in the film.

The surface tension of a solution is in general notably different from that of the pure solvent, and in case of water, whose surface tension is the greatest of any liquid with which we are concerned, even a minute trace of impurity is sufficient to lessen its surface tension considerably.

Consider a film of water stretched on a vertical ring of wire. If the surface tension remained constant, as it does in the pure liquid when the thickness exceeds 0.000001 cm., the weight of the lower part would stretch down the upper part until it broke. If the water contains some component whose depletion at the weaker points increases the surface tension, equilibrium will be preserved. In the stretching of a film, and in the general running away of the liquid between the surfaces of the film, which reduces the total available amount of the contaminant, such decrease of the concentration and increase of surface tension does occur, and the film remains stable under a considerable variation of external conditions. The formation of bubbles as a result of this variation of surface tension alone is well exemplified by a simple aqueous solution of soap or of acetic acid. The running out of the liquid between the two surfaces is greatly retarded by the viscosity of the liquid, a property which may be largely influenced by the surface adsorption of one or more of the components.

When, then, gas bubbles are introduced into a liquid pulp where oil is present there is formed about each bubble a liquid film whose surface tension is less and whose viscosity is greater than that of the bulk of the liquid. Some of the solid particles of the pulp move into the film and are raised to the surface with the bubble. Since there is a concentration of oil in the film, and since the diminution in potential energy at an oil-sulphide contact is greater than at a water-sulphide contact, the contaminated layer replaces the water on the sulphide surface and the

sulphide moves into the bubble film, while the gangue, on which water displaces oil, remains in greater measure in the body of the pulp. The bubbles, therefore, as they arrive at the surface, carry an excess of the sulphide minerals. Upon their arrival at the surface, the bubbles of the contaminated liquid persist, owing: (1) to their lower surface tension; (2) to their ability to adjust this tension to a state of stable equilibrium; and (3) to their greater viscosity which is markedly increased by the presence of the solid particles.

Mechanical-Agitation Froth Process

Sulphide and gangue minerals are beaten up with water and oil, with or without acid, then allowed to flow into a box containing a considerable body of liquid nearly at rest. Bubbles coated with a preponderance of sulphide particles float to the surface and form a heavy, persistent froth. The gangue particles sink.

Two cases arise:

Case 1.—Sulphide, gangue, water and oil.

Case 2.—Sulphide, gangue, water, oil and acid.

Case 1. *Sulphide, Gangue, Water and Oil.*—When this pulp is beaten, air is mechanically entrapped in the form of bubbles. At the surface of every bubble in the mass there is a gas-contaminated-liquid contact which results in the adsorption at this surface of the contaminant, oil, and the production of a viscous film into which the sulphide particles, circulating in the mass, pass with a diminution in the potential energy of the system. The result is that in a very short time after the air bubble is entangled in the pulp, it is surrounded by a viscous sheath composed of an oil-water interfacial film in which are entangled a large number of sulphide particles. The presence of the solid particles greatly increases the viscosity of the bubble sheath. When the solid-coated bubble arrives in the settling box or spitzkasten it rises to the surface. Here the bubble persists, or, bursting, transfers its load to other bubbles. This bubble persistence is due to a combination of several factors. The oils used have, in general, a slower evaporation rate than water. The tension of the bubble film is lower than the tension of a pure water bubble. The bubble has the power of adjusting itself to its tension, within limits, without bursting. The presence of the large amount of solid matter enormously increases the viscosity of the film.

Case 2. *Sulphide, Gangue, Water, Oil and Acid*—The addition of acid has the twofold effect of further lowering the surface tension and increasing the adhesion ratio $\frac{\text{oil-solid}}{\text{water-solid}}$. The result is, in general, cleaner concentrate with or without an increase in the sulphide content of the tailing.

Heating the pulp has, in some cases, a beneficial effect. Where this is true it is probably due to (*a*) decreased surface tension and consequent increased stability of the froth; (*b*) increased number of air bubbles formed by the air released from solution; (*c*) in the case of viscous oils, the greater area over which oil is spread and consequently the greater number of bubbles with a viscous oil-water interfacial film; (*d*) probable increase in solubility of the oil and consequent greater diffusion, resulting in more widespread adsorption in bubble films.

Pneumatic Froth Process

Sulphide and gangue minerals mixed with water and oil, with or without acid, are run into a tank with a porous bottom through which air is forced. The air bubbles rise to the surface with a coating of solid particles, preponderantly sulphide, while the gangue particles sink.

The principles involved in this method are the same as explained in the agitation-froth process. The only difference is in the method of introducing air. The result of this difference is that the bubbles in the pulp are much larger than in the agitation froth method; they arrive at the surface less heavily loaded in proportion to their area; the bubble films are, therefore, less viscous, and the froth less persistent.

Potter-Delprat Process

Sulphides and carbonates, with or without other gangue minerals, are treated with hot, dilute sulphuric acid. Bubbles of carbon dioxide and hydrogen sulphide are formed which rise to the surface with a sulphide coating and there form a froth. That part of the gangue not dissolved remains immersed. In this method, as in the other froth methods, gas bubbles are formed which are surrounded by films of contaminated water, the contaminants in this case being sulphuric acid, lead sulphate, calcium sulphate and other salts formed by the action of the sulphuric acid. The films have a higher viscosity and a lower surface tension than is possessed by the bulk of the liquid. The sulphides move into the bubble films because the system composed of sulphide and this contaminated layer has a lower potential energy than the system composed of the pulp in the bulk of the liquid. The writers at first suspected that the selective action in this case might be due to preferential gas adsorption at the gas-sulphide contact, as opposed to a gas-gangue contact, but microscopic examination of mineral froths collected from the process showed that the solid particles in the froth were completely within the films and at no point in contact with gas. The persistence of the froth is due to the factors explained in connection with the other froth processes.

While the writers have made no appeal to electrostatic forces or to

colloidal phenomena in this discussion, they realize that the potential energy existing at the contact of dissimilar substances may well include electrical forces and that migration of the suspended solid particles under the influence of electrical charges, similar to the migration of colloids, may account for some of the selective action of the bubble films. But the agitation of the pulp in the mechanical- and pneumatic-froth processes and the generation of carbon dioxide gas on carbonate particles in intimate contact with sulphides in the acid-froth process, are sufficient, in their opinion, to bring every sulphide particle into contact with a bubble film. Once in contact, the preferential adhesion of the contaminated layer to a sulphide surface in the presence of water is sufficient to account for the persistent attachment of the sulphides to the bubble films, while on the other hand, the replacement of gas or oil by water on the surface of gangue particles explains the wetting and continued immersion of the latter.

The writers have a considerable bulk of experimental data on which many of the statements in the foregoing explanation are based. These, together with the data from other experiments which are projected, and photographs of many of the phenomena mentioned, they hope to present in a later paper.

Discussion

Oliver C. Ralston, Salt Lake City, Utah (communication to the Secretary*).—This paper has appealed to me as being one of the most lucid, well-connected and complete papers on this subject which has been published. To be sure, the concepts called upon in an explanation of the flotation process by these authors are mostly old ones which have been more or less discussed in previous literature, but they have nevertheless been assembled in a most constructive and suggestive manner. We find a new contribution to flotation theory in the effect of fine particles increasing the viscosity of the interfacial film between a liquid and a gas. This phenomenon has been variously hinted at but never definitely stated in accepted scientific terms.

The discussion of "contact angles" is very clear and will doubtless be welcomed by those who have not had the opportunity to investigate the literature on this subject, but it is unfortunate that the "hysteresis of the contact angle," observed by Sulman and reported some years ago, is not mentioned or explained.

A few other points need discussion in order to call attention to the fact that, while this paper is the best discussion of the subject which has yet appeared, all of the truth has not yet been told.

On page 559, in discussing the reasons why heating a pulp often allows the recovery of a higher-grade froth concentrate, four different factors are mentioned. One of these factors deemed probable by the

* Received September, 1916.

authors is that there is an "increased number of air bubbles formed by the air released from solution." While it is commonly known that heating a solution expels the dissolved air or other gases, to one who has seen the great volumes of air that enter the pulp in a flotation machine it is hard to believe that the pitifully small number of additional air bubbles released by heating the solution can have such a good effect. I would call attention to the following more probable explanation of this fact: the viscosity of water at 80°C. is less than one-fifth of the viscosity at 0°C. In other words, on heating an ore pulp the fluidity of the water in the pulp is very greatly increased and much less water containing entrained gangue will be carried along with the froth during flotation. In other words, a cleaner concentrate is obtained. The surface tension and other physical properties of water are not altered in anywhere near the same degree by change of temperature, and I would suggest that the viscosity effect is probably responsible for the major portion of the improvement.

I notice that in their discussion of the Potter-Delprat process the authors claim to have observed microscopically the condition of the solid particles in the froth, and find them completely "within the films and at no point in contact with gas." The manager of one of the large Australian companies using flotation told me that two physical chemists in his employ consumed about 6 months' time in an attempt to definitely find out whether the mineral particles were completely submerged in the film or whether they were at some point in contact with gas. In view of their failure to reach a decision, I am inclined to ask the present authors if they are certain of this observation.

Further, we notice that electrical forces are not called into question in this discussion of flotation, although the authors acknowledge that the potential surface energy in the contact surfaces between dissimilar substances might well include electrical forces which might be called on in explanation of certain aspects of flotation. However, they feel that "*Once in contact* the preferential adhesion . . . to a sulphide surface in the presence of water is sufficient to account for the persistent attachment of the sulphides to the bubble films." I feel that they are justified in this opinion, but having seen particles migrate under the microscope when in an electrical field, and knowing that the various kinds of particles and bubbles carry electrical charges, may I not suggest that the electrical charges assist the various finely divided phases to *get in contact?* The authors consider that mechanical agitation, etc., is sufficient to get the various phases into contact, but the acceptance of these electrical phenomena in no way damages the theory which they have presented and merely adds another contributing factor to the theory.

Finally, the authors say that they have not appealed to colloidal phenomena in this discussion. This statement, in the face of the fact

that the largest single section of the paper is headed "adsorption," and that most of the forces called upon are those of "*capillarity*," is hard to understand. While I agree with the authors almost entirely, in so far as they go, I find that their paper is filled with something very closely akin to colloid chemistry. The study of capillary forces comes under the head of physics and the study of the effect of chemicals on these capillary forces comes under chemical physics or physical chemistry. Now the physical chemistry dealing with capillary phenomena is known as colloid chemistry, and Freundlich even went so far as to entitle his text on colloids, *Kapillar Chemie*. So it can be seen that while I am delighted with the work these authors have done in formulating their theory, I am disappointed in the way they have attempted to name it. I would say that it can be accepted as part of a *colloidal theory* of flotation.

A New Flotation Oil

BY MAXWELL ADAMS, RENO, NEV.

(Arizona Meeting, September, 1916)

CONSIDERABLE interest has recently been developed in sage-brush oil because of its possible utilization as a flotation agent in the mining industry. A list of some of its physical properties, together with the method used in its extraction, may prove of interest at this time.

Something over a year ago, a study of the essential oils in desert plants was begun in the Chemical Laboratory of the University of Nevada. None of the oils so far studied possess properties of special interest to engineers, except the oil of sage, *Artemisia tridentata*, which has exceptional power as a flotation agent. This plant, known as common sage brush, also called black sage, is widely distributed over the semi-arid West, being found quite generally on most of the dry plains and mountains west of Missouri.

The method of extracting the oil followed in these experiments is very simple. The leaves, twigs and small branches are placed in an air-tight drum, having a capacity of about 27 cu. ft. Steam is admitted through a number of small openings at the bottom of the retort, and the pressure maintained at 20 to 25 lb. per sq. in. for 3 hr. The escape of the steam from the retort is regulated by allowing it to pass through a stop-cock into a condenser. The water in the receiver is drawn off from time to time and the oil, which is insoluble and floats upon the water, is thus collected. At the end of 2 hr. most of the oil has been driven out, though traces continue to come over for a much longer time. By raising the pressure, the time required could probably be shortened and the yield increased, but the lack of laboratory equipment has prevented the carrying out of this experiment.

The stock wood, bark and branches contain no oil, the distribution of the oil being limited to the leaves and young shoots. There is a seasonal variation in the amount of oil contained. Samples collected on different dates gave the following amount of oil: May 1, 0.42 per cent.; May 27, 0.6 per cent.; June 30, 0.72 per cent.; Aug. 1, 0.9 per cent.; Sept. 10, 1.0 per cent. The increase appears fairly constant from early spring, when the leaves first appear, until light frosts occur in the autumn. When the plant is air-dried there is some loss of oil, as the following data will show: Two 100-lb. samples were collected at the same time. One was distilled when green; the other was air dried for

10 days before distillation. The green sample yielded 275 grams, and the dried sample 248 grams of oil, showing a loss of about 10 per cent.

A laboratory experiment can furnish few data useful in forming an estimate of the commercial cost of production. A man working for 6 hr., and using a pair of common pruning shears, collected twigs which yielded 1 lb. of oil. Since only a small percentage of the oil is lost if the brush is dried, the most economical method of production would perhaps be to collect it in large quantities, by using a tractor engine and a drag, in some such way as land is cleared for farming. When the brush is dry, the leaves and young shoots are easily shaken from the limbs. Thus the amount of material to be distilled would be greatly diminished and the oil perhaps obtained at a cost and in quantity sufficient to make it available as a flotation oil, if not alone, possibly as an ingredient, to increase the flotative power of other oils.

The crude oil is dark in color. When redistilled with steam it is water-white at first, changing gradually to a straw-yellow color upon standing. It has the following physical properties: Density at 15°C., 0.9206; refractive index at 20°C., 1.4732; rotation at 20°C., −4.69. At 98°C., a light oil, with a very sharp and pungent odor, begins to distill, but only after the temperature is above 165°C. does rapid distillation take place. At 180°C., the oil turns dark and decomposition begins. At a pressure of 12 mm., and below 125°C. almost all the oil can be distilled.

The chemical properties of the oil are as yet undetermined. There are small quantities of alpha and beta pinene. The main part of the oil has a camphor-like odor and taste, but has failed to give the ordinary tests for ketones. The fraction boiling at 175° to 180°C. has some of the properties of ordinary cineol, but is acted upon by metallic sodium, which indicates that the chief ingredient is not cineol. The chemical composition, which has little interest in this connection, will be worked out later. The important question for the engineer is: Can the oil be produced in quantity and at a cost that will make it available for ore flotation?

Discussion of this paper on p. 573.

A New Source of Flotative Agents

BY G. H. CLEVENGER, PALO ALTO, CAL.

(Arizona Meeting, September, 1916)

THE reagents now used in flotation consist of various acids or salts, which may be either electrolytes or non-electrolytes, dissolved in water and some substance or combination of substances which function as collecting or frothing agents. At times the only dissolved salts present are those naturally occurring in the water used. The general effect of the electrolyte is to greatly sharpen the separation between the gangue and the concentrate. Examples of this are: The use of sulphuric acid with zinc ores; and of sodium carbonate or calcium oxide (lime) with silver-gold ores. Crude pyroligneous acid is also sometimes used when avail able. Various oxidizing agents, such as permanganates, bichromates, etc., are added in the selective flotation of lead-zinc ores. Many other reagents for performing certain specific functions, either real or imaginary, have been proposed, and a number of them have been tried upon a working scale. The wild orgy of experimentation which is now going on in flotation exceeds even that which followed the introduction of the Washoe process for the treatment of Comstock ores, when, among other things, sage-brush tea and tobacco juice were reagents added to the pans. Out of all this will, of course, eventually come a more or less standard practice for the treatment of each class of ore.

Omitting a discussion of the functions of frothing and collecting agents, the reagents used in flotation for this purpose may be classified under five general heads: (1) Essential oils; (2) fixed or fatty oils; (3) alcohols and their combinations with organic acids; (4) coal tar and its refined products; (5) petroleum and its refined products.

A flotative agent of some kind is required in flotation as now practiced. A single reagent, as, for example, certain of the essential oils, may perform the dual function of frothing and collecting agent, or a mixture of substances may be required.

In this country, the essential oils used have been wood products, and have been almost exclusively confined to the steam-distilled pine oils. At the present time, the supply of these is limited, and the cost almost prohibitive, so that their use has been dispensed with as far as possible. The oils and tars resulting from the destructive distillation of

the various species of the coniferæ find a more general use on account of the greater supply and lower cost, although even these have risen in price and the future supply is somewhat problematical if the demand for them in flotation continues to increase at the same rate as in the past.

The fixed or fatty oils, with the exception of oleic acid and crude pyroligneous acid, are scarcely ever used, since they usually give poor results as compared with other flotative agents.

Coal tar is a common and cheap product, but not all coal tars are suitable for use in flotation. Furthermore, the marketing of coal tar is coming more and more into the hands of large distributors, who contract for the output of the various gas plants. The refined products of coal tar are useful flotative agents, but in the case of certain of these, for example, carbolic acid and cresol, the price at present has become prohibitive. In general, crude coal tar yields the best results when mixed with other oils. It is a frequent practice to use a small proportion of pine oil, in order to modify the froth. Fortunately, in many cases these cheaper mixtures yield very satisfactory results, and can now be obtained at a reasonable cost.

It appears that only the crude petroleum oils having an asphaltic base are generally useful in flotation. These are invariably mixed with other oils. The refined products of petroleum are usually not satisfactory, except as constituents of oil mixtures. The one exception to this is perhaps the case of kerosene acid sludge, which is a byproduct of petroleum refining. Kerosene acid sludge from eastern refineries gave unsatisfactory results at Anaconda, but California kerosene acid sludge, resulting from the refining of crude oils having an asphaltic base, is satisfactorily used in conjunction with a small proportion of wood creosote for the treatment of Anaconda copper ores. While California kerosene acid sludge is a satisfactory flotative agent with many ores, the available supply tends to limit its use. Formerly California refiners threw it away, but now little of it is available upon the open market, since practically the whole of the present production is contracted for a long time in advance by pioneer users.

A suitable oil supply, both as regards the character of oil to give the best results and its present and future availability, is a matter of serious consequence to companies operating flotation plants. It was with this in mind that I began the investigation of possible sources of flotative agents, with the particular object of affording relief to mines located in the arid regions of the West, where none of the common flotative agents are locally available, and where transportation costs upon those from the outside are high.

Certain plants and shrubs have the peculiar property of secreting oil in the new growth, during the growing season, and particularly in the leaves. Botanists appear to be uncertain regarding the function of this

oil in the metabolism of the plant. Some hold that it is a reserve food supply for the plant, while others believe that it is waste product which the plant fails to throw off. Be this as it may, many plants, shrubs and trees contain oil in the leaves and new growth. A good example of this is the case of the eucalyptus tree, from the leaves of which essential oils, which are used for various purposes, are recovered by steam distillation. The leaves of the variety known as amygdalina, unique as being the tallest tree in the world, have afforded a large proportion of the flotative agents used in the concentration of complex Australian ores. This variety of tree is fortunately very plentiful in close proximity to these ore deposits.

In the great arid and semi-arid mining regions of the West, the most common of the few plants and shrubs native to the region are the varieties of sage brush known as mountain sage, pasture sage, wormwood sage, etc.; also, in certain regions, greasewood and other shrubs of a similar nature. It is, therefore, to sage brush that my attention has been directed in the search for flotative agents for concentrating Western ores.

It appears that the various varieties of wild sage were first investigated by certain members of the Department of Agriculture, with regard to the possibility of producing from them by steam distillation essential oils suitable for pharmaceutical use. Steam-distilled oil was prepared by Rabak[1] from the variety *artemisia frigida* in 1905, and during the summers of 1907 and 1908 larger quantities of oil were prepared from specimens of this plant, collected in South Dakota. In 1912, an essential oil prepared by the steam distillation of *Ramona stachyoides* (black sage) from southern California was also reported by Rabak. This oil was said to contain 40 per cent. of camphor. In 1914, Charles E. Burke and Charles C. Scalione[2] gave an account of an investigation of an essential oil which had been prepared from the same shrub by the steam distillation of several hundred pounds of leaves and twigs collected from brush growing near Riverside, Cal. The yield of oil in this case was 0.9 per cent. of the weight of the material used. This yield was somewhat higher than that reported by the Department of Agriculture. This is perhaps accounted for by the fact that the brush was collected later in the season. This oil is reported as having the following composition:

	Per Cent.
Pinene	6
Cineol	30
Dipentine, terpinene, etc.	25
Thujone	8
Camphor	25
Resinous material	5

[1] Frank Rabak: Wild Volatile-oil Plants and Their Economic Importance, *Bulletin* No. 235, p. 22, *Bureau of Plant Industry*.

[2] Charles E. Burke and Charles C. Scalione: Investigations on Oil of Black Sage, *Journal of Industrial and Engineering Chemistry*, vol. 6, p. 804 (1914).

Camphor was separated from a portion of the oil, thus demonstrating the rather interesting possibility of black sage as a source of borneol camphor. Upon request, Mr. Scalione furnished me with a small sample of the original oil. This oil is clear, with a very slight yellowish tinge, and has an agreeable odor. In fact, so far as appearance and general behavior goes, although the chemical composition is somewhat different,

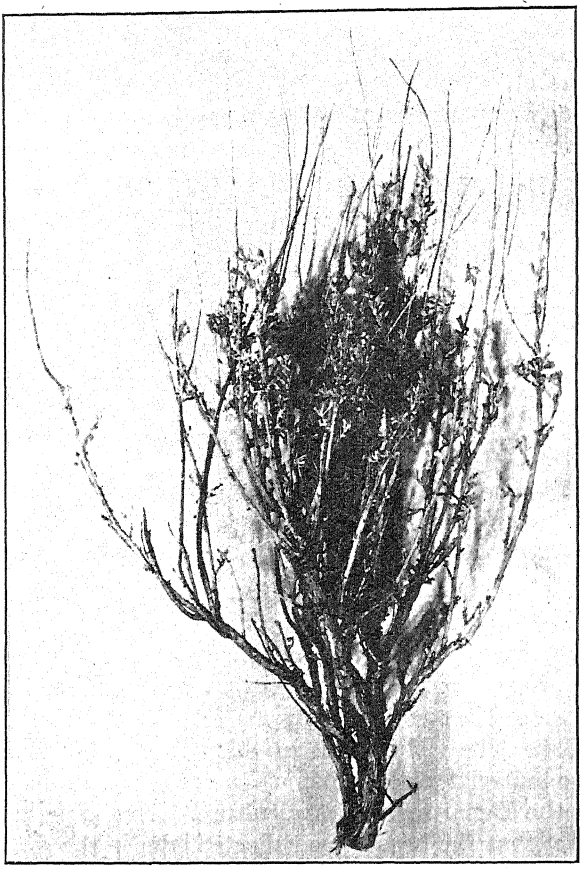


FIG. 1.—ARTEMISIA TRIDENTATA.

it very much resembles steam-distilled eucalyptus oil from the Australian variety, *amygdalina*. This oil is a good frothing agent, and it yielded quite satisfactory results upon lead and zinc ores in a qualitative way, although the amount available did not permit of thorough investigation. The idea of investigating sage brush and greasewood as possible sources of flotative agents was conceived early in January, but it was not until early in March that the first 100 lb. of sage brush was forwarded to me,

through the courtesy of G. B. Lantz. This lot was collected near Goldfield, Nev., and proved to be the variety *Artemisia tridentata.* "In the Western Arid Transition zone the flora consists largely of the true sage brush, *Artemisia tridentata,*"[3] therefore this would be the variety available in the greatest abundance near to the mines of this region (see Fig. 1).

An apparatus for destructive distillation, capable of treating 30 lb. of brush at a charge, was constructed, and two charges of the brush were distilled (see Fig. 2). The products which first came over consisted of acid liquor resembling the crude pyroligneous acid obtained from wood distillation, a black oil or tar, and inflammable gas. Finally these products ceased to come over, but, upon raising the temperature, a consider-

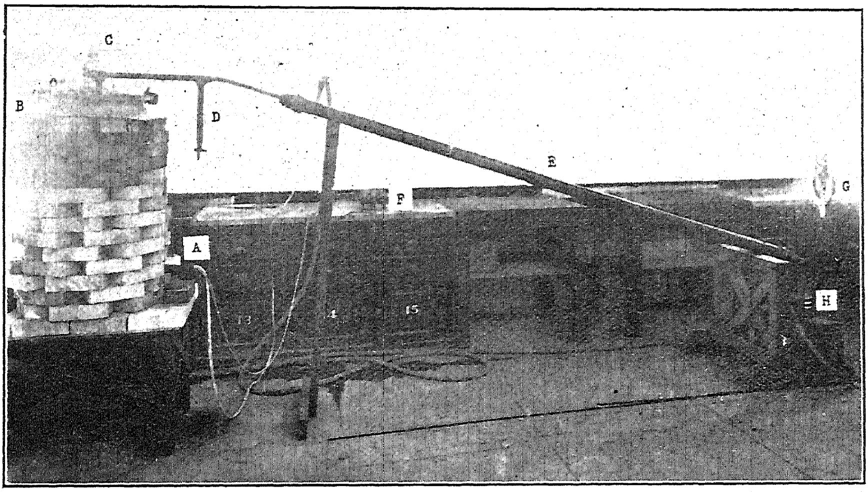


Fig. 2.—Experimental Apparatus Used for the Destructive Distillation of Sage Brush. A, burners; B, retort; C, cold junction; D, tar tap; E, condenser; F, galvanometer; G, gas; H, distillate.

able amount of gas was given off and a rather thick reddish-brown liquor, having an alkaline reaction, began to come over, and with it a small amount of tar similar to that which came over with the acid liquor. The brown liquor had the characteristic fishy odor peculiar to the amines. There was also at times a distinct ammoniacal odor. The acid and alkaline liquors were kept separate, while the tar from both was combined. These three products were first tried qualitatively in the flotation of finely ground samples of various minerals such as galena, cinnabar, pyrite, etc.

The acid liquor behaved very much as does ordinary pyroligneous acid. The alkaline liquor was a good frothing agent, but the froth

[3] *Encyclopædia Britannica,* Eleventh Edition, vol. 27, p. 634.

carried up little mineral. The tar which came over with the acid liquor proved to be a splendid flotative agent.

Following this, quantitative tests were made upon a number of different ores, employing the tar produced from this lot of sage as a flotative agent. These results, a number of which are given below, were in general very satisfactory.

A sample of zinc ore from the Butte Superior mine, Butte, Mont., containing 22.39 per cent. of zinc, when tested in a Janney laboratory machine employing a solution of 0.25 per cent. of sulphuric acid, gave 97.0 per cent. extraction of the zinc. The first concentrate contained 53.8 per cent. of zinc, the second 48.9 per cent., the third 40.5 per cent., and the fourth 18.1 per cent. The oil consumption was at the rate of 0.4 lb. per ton of ore.

A sample of mercury ore from the New Almaden mine, California, containing 0.26 per cent. of mercury, when tested in a Janney machine employing a 0.2 per cent. solution of sodium carbonate, gave an extraction of 90 per cent. of the mercury. The first concentrate contained 3.6 per cent. of mercury, the second 2.5 per cent., the third 1.55 per cent., and the fourth 0.95 per cent. The oil consumption was at the rate of 1 lb. per ton of ore.

A sample of lead ore from the Coeur d'Alene region, containing 12 per cent. of lead, when tested in a Janney machine employing a 0.05 per cent. solution of sodium carbonate, gave an extraction of 92.2 per cent. of the lead. The calculated lead content of the total concentrate was 37.3 per cent. The oil consumption was at the rate of 0.67 lb. per ton of ore.

A sample of silver-gold ore from the Ophir mine, Virginia City, Nev., assaying 0.46 oz. gold and 7.4 oz. silver per ton, when tested in a Janney machine employing a 0.1 per cent. lime or sodium carbonate solution, gave an extraction of approximately 90 per cent. of the silver and 95 per cent. of the gold. The first concentrate assayed gold 5.3 oz. and silver 198 oz. per ton; the second, gold 3.75 oz. and silver 72.9 oz. per ton; and the third, gold 1.32 oz. and silver 35.3 oz. per ton. The oil consumption was at the rate of approximately 0.6 lb. per ton.

The oil consumption when employing sage tar appears to be less than with most of the other oils experimented with in treating the same ores, and the extraction was in general better. Since it is the experience of many that large-scale operation requires less oil than is indicated by small-scale tests, it is reasonable to suppose that the oil consumption in treating the ores cited would be materially lessened when working upon an operating scale, and that possibly the extraction could be bettered. I think the latter is particularly true of the Ophir ore.

Later in March, another lot of brush of the same variety was collected by Mr. Lantz from the same locality, more fully in leaf than that collected

earlier in the season. Distillations were run on this lot, keeping the various products separate, so that they could be properly measured and weighed.

Temperature measurements were made at regular intervals by means of a thermocouple, and as a result there was better control of the heating than in former tests. The yield from the last of these tests, which was the most carefully conducted, and was therefore the most representative, was as follows:

	Pounds	Per Cent
Weight of sage brush used	30.	
Acid liquor	8.4	28.00
Tar with acid liquor	1.1	3.66
Alkaline liquor	0.66	2.20
Tar with alkaline liquor	0.09	0 30
Charcoal	10.00	33 33
Gas (by difference)	9 7	32.51

The retort was slowly heated for 1 hr. before liquid began to condense. The temperature at the center of the charge at the end of this time, as indicated by the thermocouple, was 60° C.; the temperature at the sides of the charge was probably somewhat higher.

In the next period of 6 hr., during which the acid liquor and most of the tar came over, the temperature rose from 60° to 275° C.; the rise above 100° C. taking place during the last hour and a half.

The alkaline liquor and the last of the tar came over in the last period of 3 hr., during which the temperature rose from 275° to 611° C.

It is reasonable to assume that a yield of about 4 per cent. of the tar oil can be realized if the sage is collected at the proper season and the distillation carried on by the best methods. Then there is the acid liquor, which in certain cases could be used directly in flotation; or it might prove profitable to recover the alcohol, acetic acid, the dissolved tar oil, etc., which it contains. I have not had an opportunity to investigate the alkaline liquor thoroughly, but it would seem to present many interesting possibilities. Among other things, the first lot was found to contain 2.09 per cent., and the second lot 2.18 per cent. of nitrogen.[4] I suspect that this liquor may also contain phenolic bodies which would be useful in flotation when separated from the other constituents. The charcoal is fine, but it should be possible to utilize it for fuel in heating the retorts. One peculiarity is the high percentage of ash which it contains (10.5 per cent. in the one sample analyzed). This may be in part due to dust upon the sage brush, although the brush was chopped fine before distillation, and it would seem that a good deal of the dust would be shaken from it during this operation. If the charcoal were burned, as previously suggested, the alkaline ash might serve instead of lime or sodium carbonate in cases where flotation in alkaline solution was practiced.

[4] Analysis by Professor R. E. Swain.

The inflammable gas would, of course, be burned under the retorts The proportion of the heat necessary for carrying on the operation which could be realized in this way is problematical; however, it is reasonable to assume that a considerable part of that required for destructive distillation could be produced by burning the gas and charcoal. Moreover, another interesting possibility in connection with the heating of the retorts is the utilization of the waste heat from various metallurgical operations.

Through the courtesy of Charles Scalione, several ounces of steam-distilled essential oil was prepared from the tip ends of a portion of the last lot of sage brush. Approximately 1 hr. was required for distilling each charge of 20 lb. The yield amounted to 0.43 per cent. of the whole plant. This oil had a greenish-yellow color when first distilled, but became yellow upon standing. It has the characteristic penetrating odor of sage, and a very decided tendency to creep up the sides of the glass containing vessel. This oil appears to have distinctly different properties from the essential oil resulting from the steam distillation of the black sage. It gives promising results with some ores, but in my opinion it is not nearly so good as the tar oil resulting from the destructive distillation of the same shrub. In fact, under ordinary circumstances, I do not think that the steam-distilled oil can be given serious consideration, since the mines that would be most benefited are generally located where fuel is high, and the yield of oil is rather small, probably less than 1 per cent. In addition, there are no other valuable products, unless the tannin extract resulting during steam distillation should have a market value or a demand should arise for "sage tea."[5]

Time has not been available for making analyses and a study of all the products resulting from the destructive distillation of sage. However, sufficient work has been done to show at least that the light tar is an efficient flotative agent for a considerable number of ores, as indicated by the tests cited.

The idea of utilizing sage brush in metallurgy is by no means new, as is shown by the following quotation,[6] referring to early development of the Washœ process, and the wild riot of experimentation accompanying it:

"The native sage brush, which everywhere covered the hills, being the bitterest, most unsavory, and nauseating shrub to be found in any part of the world, it was not long before a genius in charge of a mill conceived the idea of making a tea of this and putting it into his pans. Soon, the wonders performed by the sage-brush process, as it was called, were being heralded through the land." Discussion of this paper on p. 573.

[5] Patents are pending covering the use of the various sage products in flotation.

[6] Dan De Quille (William Wright): *The Big Bonanza*, pp. 138, 140, American Publishing Co., Hartford, Conn., 1877.

A New Flotation Oil and A New Source of Flotative Agents

Discussion of the papers of

MAXWELL ADAMS, (p. 563), AND G. H. CLEVENGER, (p. 565).

OLIVER C. RALSTON, Salt Lake City, Utah.—We are indebted to these two gentlemen for the work which they have done with the oils derived from sage brush. Their work should not be confused, however, as the work of Mr. Adams deals with steam-distilled oils and that of Mr. Clevenger with destructively distilled oil, largely. In other words, Mr. Adams' oil is probably the original essential oil that existed in the brush before distillation while Mr. Clevenger's oil not only contained this essential oil but also some products of destructive distillation of the wood constituents.

I had noticed the work of Mr. Adams on attempting to separate the various constituents in the oils of some of the Western conifers and had sent to him to ask if he could send me small samples of some of these pure products. This he very kindly did, and with them included some samples of oil from the common sage brush, whose characteristics he has here dsecribed, as well as a sample of oil from the common "rabbit brush." Only the sage brush oil proved of much value for flotation, but it was very powerful, and I so informed Mr. Adams. Even in amounts as small or smaller than $\frac{1}{10}$ lb. per ton of ore, it gave very quick and positive flotation of many ores, with high-grade concentrates and good extractions. It proved to be especially desirable for the flotation of carbonates of lead and of copper which had received an artificial coating of sulphide by treatment with a solution of a soluble sulphide.

I also received a sample of crude pyroligneous acid from sage brush prepared by Dr. W. C. Ebaugh, of Salt Lake City. It did not do very good work in flotation, but it is possible that by concentration of this product to obtain the residual tar a good flotation oil might result. This has not been done as yet and I am sorry to see that Mr. Clevenger has not done this. It might be that the dissolved tar, added to the settled tar, would make a total yield of flotation oil higher than the 4 per cent. which Mr. Clevenger feels is a safe estimate. The importance of this point will be seen in the following discussion of the possible costs of sage brush oils for flotation.

I have been fortunate in obtaining the approximate costs of collecting sage brush, as well as the costs of operation in the ordinary wood-distillation plants. I was given the former by E. H. Snyder, who with a number of associates cleared some land of sage brush and made potash from the ashes of the burned brush. The latter were given me by R. C. Palmer, formerly of the Forest Products Laboratory of the Department of Agriculture.

The clearing of land of its sage brush is ordinarily accomplished by means of a tractor pulling a special frame made up of railroad rails. This breaks off the bushes so that they can be collected in hay ricks and

hauled to a central point. For merely uprooting the plants and burning them on the spot the cost is about $4 per acre, but to recover them and take them to a central point ready for burning or distillation, costs about $6 per acre.

Mr. Snyder further informs me that with the average 4-ft. stand of brush in southeastern Nevada the yield per acre is about 7 tons of brush. I am informed that there are considerable stretches of country around Bear Lake in Idaho and Utah where the brush is nearly 10 ft. high. Hence considerably higher yields might be possible. However, we can take as a safe figure $1 per ton for cutting the brush and carrying it to a central point.

The cost of destructive distillation of hard wood is in the neighborhood of $8 per cord (4,000 lb.) of wood, or about $4 per ton—possibly $5 per ton of wood. While the sage brush might be more bulky than the wood, and hence cause higher costs, this factor can not be estimated at the present time and it is probably best to count on a distillation cost of $5 per ton.

That would make the total cost of treatment of each ton of sage brush about $6, and the yield, according to Clevenger, is 4 per cent., or 80 lb., of tar. This would mean about 10 gal. produced at a cost of $6, or $0.60 per gallon, or 7.5 c. per pound. This is rather high but is comparable with the present price of pine oil, of which sage brush oil seems to be the full equivalent, if not the superior. With most ores, less than 0.5 lb. of sage brush oil should be needed per ton of ore.

If Clevenger's figure of 4 per cent. yield of oil could be raised to 6 per cent. by addition of the dissolved tar from the pyroligneous acid, the cost of the oil would be about 43 c. per gallon. Further, if greater yields of brush from each acre were obtainable, the costs would be cut slightly, although this point is not of so much importance, as the principal costs are due to the distillation. It is also possible that in some cases owners of the land would be glad to pay $1 to $2 to have their land cleared of brush, where it would usually cost them about $4. Only skillful management and the choice of the proper regions for operations would probably bring in this latter source of income.

There is one other source of income from the products of the sage plants, namely, the potash. Mr. Synder informs me that he found the potash in carefully burned brush ashes to amount to 15 to 20 per cent. of the total weight. Most of this was soluble in water but the remainder was "acid soluble K_2O." The ash amounts to 7 to 10 per cent. of the weight of the plants. Hence the K_2O content of the brush can be estimated as from 1 to 2 per cent., or 20 to 40 lb. per ton of brush. In normal times potash is worth about 3 c. per pound. This would make the value of the potash in the brush about $0.60 to $1.20 for every ton. The cost of leaching and crystallizing the potash from the ashes is

unknown, but might possibly be $3 on every ton of ash or 20 to 30 c. referred to every ton of brush. It is not impossible that the burning of the sage brush charcoal under the boilers of the distillation plant, followed by leaching of the ashes would yield potash of sufficient value to allow a profit, and hence allow of production of sage brush oils for flotation at a lower net cost than the above rather pessimistic figure. One difficulty at the potash end of the work is that it tends to be carried away in the combustion gases and a Cottrell precipitator might have to be used in order to insure recovery of all the potash.

The above is merely a suggestion to show what might be expected if one were to attempt the commercial production of sage brush oil. Rather high costs were assumed, but it is certain that other sources of expense than those enumerated will make such a pessimistic figure desirable, for the sake of safety.

About 10,000 gal. of steam-distilled pine oil are being used every month in the United States for flotation purposes and more would be used if the price were lower. The last quotation that I heard for pine oil was 67 c. per gallon, f.o.b. New York. If sage brush oil could be produced for 40 c. per gallon it is probable that the market would jump to at least 25,000 gal., or roughly 1,000 gal. per day. With this would be produced 3,000 to 4,000 lb. of potash, an amount which is much less than 1 per cent. of the total consumption in the United States. Hence the sale of potash from such a source should have no bad effect on the market. On the other hand, if the oil could be produced for 25 c. per gallon, there is no reason why its use should not amount to 10 times the consumption of the oil at a 40-c. rate.

It is to be hoped that these figures are attractive enough to cause experimentation on a reasonably large scale, to determine whether operations can be profitable. Only by such a method can it be estimated whether we have a new source of flotative agents and a new source of potash.

History of the Flotation Process at Inspiration

BY RUDOLF GAHL,* PH. D., MIAMI, ARIZ

(Arizona Meeting, September, 1916)

CONTENTS

* Metallurgist in charge of Concentrator, Inspiration Consolidated Copper Co.

The history of flotation in America is very short, at least as far as the large-scale application of the process is concerned. It is remarkable how many important developments have taken place in the last few years and are already being extensively utilized. What was new a year ago, seems almost commonplace now. For this reason it is with hesitation that I follow the suggestion of Dr. Ricketts, President of the Institute, to describe the experiences of the Inspiration Consolidated Copper Co. with the flotation process.

Tests Conducted in Small Test Mill

When the Inspiration company first decided to build a concentrating plant, nothing was known about flotation, and the process was to be gravity concentration pure and simple.

Demonstration Tests Conducted by Minerals Separation Co.

While plans were being prepared by H. Kenyon Burch (who had been intrusted with the design and construction of the concentrator), the Minerals Separation Co., a concern at that time little known in America, asked and obtained permission to demonstrate the value of its flotation process for Inspiration ore. As a consequence, a small 50-ton flotation machine of standard design was added to the company's test plant and started to operate in the beginning of 1913. This marks the beginning of flotation at Inspiration.

The results obtained with this machine, which was operated by members of the Minerals Separation staff, so far surpassed what this company anticipated that it was decided to continue flotation tests for this purpose, and two of the flotation experts of the Minerals Separation Co., I. L. Greninger and G. A. Chapman, were retained. L. R. Wallace, now superintendent of the Miami works of the International Smelting Co., was at that time metallurgist of the Inspiration company and in that capacity took an active part in these tests. Great credit is due to him for his quick recognition of the possibilities of flotation.

Flotation Tests Conducted by Inspiration Co.

The tests led to the conclusion that it would be advisable to incorporate flotation into the concentrating process. Doubt existed only as to the extent to which this should be done. The first design brought out by Mr. Burch only called for flotation treatment of the concentrator tailings. While the tests were progressing, it became more and more evident, however, that flotation should form a more essential part of the milling process, and it was finally decided to leave out all the complica-

tions which are usually adopted in gravity concentration plants for the purpose of recovering as much as possible of the mineral values of the fines, and to rely on flotation alone for this purpose.

Sampling of Orebody

This decision was reached only after it had been established by numerous tests that the orebody as a whole was suited for flotation treatment. It was found, it is true, that a portion of the ore in the mine was unsuitable for flotation by the process in question, but the amount of this ore was established to be only a small fraction of the total. Besides, the tests proved that the ore contained in this fraction lost its refractory character when mixed with the rest of the ore.

Mr. Greninger gives the following description of the manner in which these tests were carried out:

"Sample lots, each amounting to about 10 tons and representing from 30 to 35 ft., were blasted from the back and sides of the drifts and after suitable crushing and grinding treated in the small flotation machine mentioned above.

"The results were very erratic, some showing good recovery with a high grade of concentrate, while others returned concentrate of a very low grade and also showed a low extraction.

"After a considerable number of tests had been made, it was found that the good results were always obtained when treating ore of a schist gangue, while poor results resulted when the gangue was the altered and kaolinized granite which forms a part of the Joe Bush orebody.

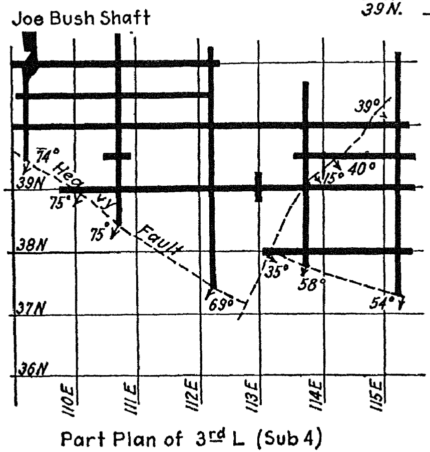
"Ten-ton samples were taken from various parts of the mine for the purpose of determining the extent and amount of this granite ore. These lots were taken from the drifts crossing the orebody from south to north, starting in the granite ore and continuing along each drift until the schist gangue was encountered.

"In all of these tests the results were uniformly good while treating the schist ore and poor while treating the granite ore.

"Subsequently, another series was inaugurated with the view of determining the amount of granite ore that could be mixed with the schist ore without interfering with the treatment of this mixed ore by flotation.

"Various percentages of granite ore were mixed with clean schist and the mixture treated by flotation. It was found that 10 per cent. of granite caused no change in the behavior of the ore in the flotation plant and when 20 per cent. of the granite was mixed with the schist ore only a slight difference was noted, this difference consisting in a slight lowering of the grade of the concentrate with a corresponding falling off in extraction."

The conditions described by Mr. Greninger are well illustrated by a drawing made by C. E. Arnold of the mine-engineering staff of the com-



Sample Nº	Assay Tails	Assay Feed (Drift $112\frac{1}{4}$ E)	Ox Cu in Feed	Samplers Description of Ground
13264	0.32	2.59	0.24	Solid Schist Perhaps a little Granite
13263	0.34	2.82	0.11	Granite and Schist, mostly Granite somewhat shattered
13262	0.35	1.80	0.27	Mostly Schist, much Quartz and little Granite
13199 (Drift 39N.)	0.26	1.94	0.21	Granite and Schist about half each
13198	0.22	3.25	0.06	Granite and Schist, mostly Schist much Quartz and hard
13197	0.22	1.70	0.01	Granite and Schist, much Quartz and hard
13196	0.32	2.82	0.24	Hard Granite and much Quartz perhaps some Schist
13195	0.10	1.42	0.03	Slabby Granite, fairly solid
13194	0.39	2.09	0.24	Slabby Granite, somewhat soft and shattered
13193	0.14	1.21	0.05	Granite slabby and solid oxidized
13192	0.17	1.67	0.18	Slabby and solid Granite oxidized
13191	0.29	1.89	0.22	Slabby and solid Granite oxidized
13190	0.42	2.18	0.19	Slabby Granite partly soft. shattered, oxidized
13189	0.31	1.59	0.12	Slabby Granite, fairly hard. oxidized
13188	0.12	1.26	0.09	Slabby Granite, shattered and soft, oxidized
13187	0.25	1.50	0.02	Slabby Granite, oxidized
13186	0.37	1.59	0.20	Shatterd Granite, soft slabby, oxidized
13185	0.68	0.82	0.02	Shattered Granite, soft slabby, oxidized
13184	0.92	1.07	0.14	Shattered Granite, soft slabby, oxidized
13183	1.21	1.40	0.53	Fault Material and Granite oxidized

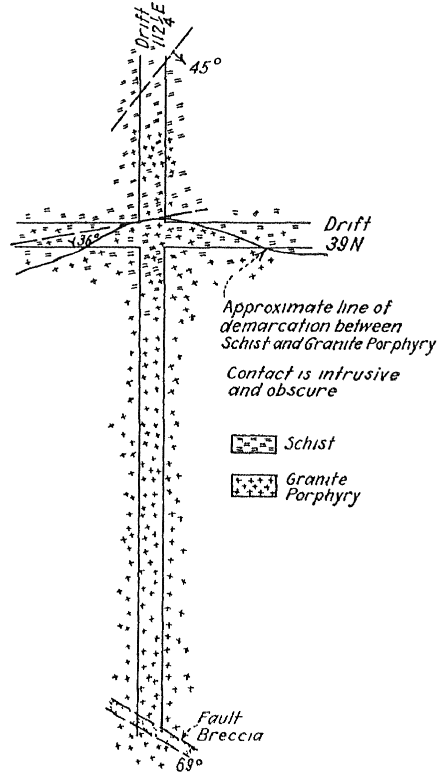


Fig. 1.—Map Showing Geology and Results from Laboratory Flotation Tests on 5-Ft. Samples Taken at 10-Ft. Intervals along Drift between 3740 and 3935 N.

pany, which represents laboratory flotation results on samples taken from a drift in the Joe Bush orebody (Fig. 1). The results show clearly that the granite by itself does not interfere with flotation, but only the fault material, evidently corresponding to what Mr. Greninger terms kaolinized granite.

Other Results Obtained in Small Test Mill

The test conducted in the small test plant established in a general way the physical conditions under which Inspiration ores could be treated advantageously; for instance, it was decided that raising the temperature did not improve the results obtained in proportion to the extra expense of such procedure.

As far as flotation oils are concerned, those in charge of the tests came to the conclusion that cresylic acid (98 per cent. pure) should be used as the main flotation agent, and should be supplemented by crude turpentine. As the most important result brought out by these tests, I consider the discovery that the flotation agents may profitably be added in the grinding machines, while it had formerly been the customary practice to add the "oils" in agitating tanks especially provided for the purpose. This discovery had an important bearing on the later developments in the Inspiration milling practice, inasmuch as it paved the way for the use of much heavier oils; for instance, coal tar which it is impossible to amalgamate thoroughly with the pulp in agitating tanks. Mr. Chapman, I think, made this important discovery during this period (U. S. Patent 1,102,874).

The details of the flow sheet to be followed were left to be decided by tests on a larger scale. Only this much was settled: That no concentration of any kind, either flotation or gravity, should be attempted before the ore was reduced to the fineness required for the flotation process. This decision was brought about by the fact that excellent recoveries were obtained when this procedure was followed, and was, of course, also strongly urged by considerations of simplicity in the milling process and cheapness of milling operations.

In these tests a 50-ton flotation machine of Minerals Separation Co. Standard type was used. In the general design, it was similar to the larger machine later put in use. It had eight agitating compartments with propellers of 12-in. diameter revolving with a peripheral speed of from 1,200 to 1,400 ft. per minute.

Tests Conducted in Large Test Mill

Flow Sheet

In order to test on a large scale the points already settled in the small test mill, and to decide the points left undecided by the small-scale experi-

ments, a 600-ton test plant was built and put into operation at the beginning of January, 1914. It was my privilege to conduct those tests in coöperation with Mr. Greninger, who represented the Minerals Separation Co., and with the representatives of other concerns, who in the course of time decided to submit flotation machines to the consideration of the Inspiration company. The flow sheet of this 600-ton test mill was extremely simple. It is illustrated in Fig. 2. The ore after being crushed to the desired fineness by machinery, which was tested out at the same time, was sent to tables; from these the concentrates went to the concentrate bins, while the tailings passed on to an 8-compartment flotation

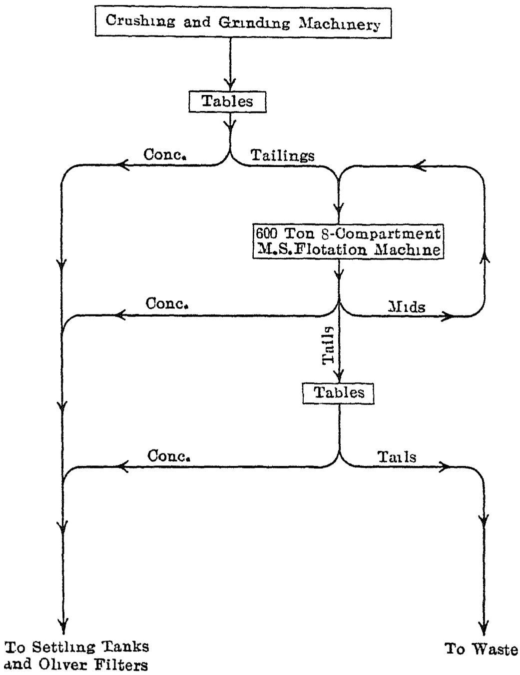


FIG. 2.—FIRST FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATED.

machine of the Standard Minerals Separation type with 24-in. stirrers. Later a 12-compartment machine of the same type was added and operated in parallel with the 8-compartment machine. The tailings from the flotation machines were passed on to other tables, the concentrates from which, combined with the concentrates from the upper tables and the flotation machine, went to the concentrate bins, while the tailings went to waste. The flotation machine was operated in the standard manner, the feed being introduced into the first agitating compartment, passed to the first spitzkasten, drawn to the second agitating compartment, sent from there to the second spitzkasten and drawn to the third

agitating compartment, etc., the agitators forming the necessary suction for the transportation of the pulp from the spitzkastens to the following agitating compartments. A final concentrate was made from one or more spitzkastens, while the concentrate from those remaining was considered a middlings product and was returned to the head of the machine for retreatment.

This flow sheet was extremely simple, but after a while even this was considered too complicated, inasmuch as the necessity of table treatment ahead of flotation treatment was doubted.

Discussion of the Value of Preliminary Table Treatment

The advantages pointed out in favor of the preliminary table treatment were about as follows: Since the tables would make a certain recovery, an impoverished flotation feed would result and assist the flotation machine to make a tailings product low in copper. However, the validity of this argument was doubted for the following reasons:

In the first place, if complications are to be avoided, the preliminary table treatment has to be of a comparatively rough character; the refinements of hydraulic classification have to be dispensed with, as by its use more water would be introduced into the pulp than would be advisable for the following flotation process. It must be taken into consideration that the pulp delivered to the tables already contains about 3 tons of water to 1 ton of solid matter, experience having shown that the grinding machines, consisting of either ball or pebble mills, deliver a product of the desired fineness, about 1 or 2 per cent. on 48-mesh, when the consistency of the overflow from drag-belt classifiers, working in conjunction with the grinding mills, was carried at about this figure; experience also showed that this consistency is suitable for the flotation treatment, while greater dilution of the pulp resulted in an increased copper loss. The logical way out of this difficulty would be to introduce settling tanks, for which, however, the modern mill designer has a just abhorrence; at least, when they are to be placed in the middle of the mill. Neither the liberal use of dressing water nor a low tonnage rate would be permissible for the same reason. On this account, a high recovery could not be expected from a preliminary table treatment. During our tests it averaged around 33 per cent.

Furthermore, the assumed fact, that an impoverished feed results in a better recovery of the flotation machine, was strongly doubted by some of the flotation experts, who claimed that in order to form a froth which had the necessary carrying power for mineral, it would be better to leave as much mineral as possible in the feed to the flotation machine. In other words, their advice was to leave out the tables in order to permit the flotation machines to do more efficient work.

There is no doubt, however, that tables will catch a certain amount of mineral (especially coarser grains) which will escape in the flotation process. For this reason, tables have to be used to insure the highest recovery,[1] but it seems that tables following the flotation treatment would make up for this deficiency of the flotation machine better than tables ahead, inasmuch as they would not work under the disadvantage, known to every mill man, of receiving too rich a feed. It is well known that it is possible to make a much lower table tailing working on feed containing a small percentage of copper than on a feed rich in copper. From theoretical speculations, therefore, no valid reason was advanced that increased recovery should result from table treatment ahead of the flotation treatment.

The next argument advanced by the supporters of a preliminary table treatment was that it would result in an improvement of the grade of the general concentrates, inasmuch as it would be possible to make a very clean product on the tables, thereby raising the general average of the concentrates. There is evidently some sound foundation for this point; how much, could not well be investigated without resorting to actual experiments.

A third point seemingly in favor of preliminary table treatment is this: Flotation concentrates offer more or less difficulty in mechanical handling; vacuum or pressure filters have to be resorted to for this purpose. It was pointed out, therefore, that if a certain percentage of the total concentrate could be saved on the tables, and a table concentrate produced containing only a small amount of the slime, so troublesome in filter treatment, it might be a decided advantage; the quantity of concentrate to be handled on filters might be materially reduced and economies effected in this manner. At the same time, experience shows that the flotation concentrate resulting under those conditions, because it contains less of the coarser sand, is more difficult for the filters to handle than it would be were the sand left in. Very likely, therefore, no decided improvement in the handling of the concentrates would result from the separation of the table concentrates from the flotation concentrates, at least under the conditions prevailing here.

The same objections do not hold true for tables when used after the flotation process, as in that case no limitations are imposed regarding the hydraulic and dressing water, and the tables cannot help but make an additional small recovery of concentrate. On account of the fact that the table feed is necessarily low in copper, the tailings will be correspondingly low. In all cases, therefore, where flotation treatment does not make a full recovery, or practically so, of mineral values, tabling after flotation seems imperative.[2]

[1] This applies only to the flow sheet under discussion.

[2] This remark again refers only to the flow sheet under discussion.

Although such theoretical considerations were indulged in, no decision was based on them. To settle the main points in question, a series of tests was carried out. On alternate days, the "upper" tables were bypassed (Fig. 3) while on the remaining days, they were utilized (Fig. 2)

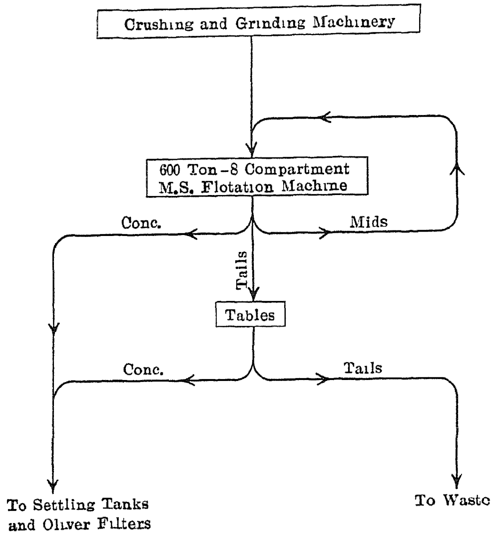


FIG. 3.—SECOND FLOW SHEET OF 600-TON TEST MILL, INSPIRATION CONSOLIDATED.

TABLE 1.—*Comparison of Efficiency of Flow Sheet No.* 1 (*Fig.* 2) *and Flow Sheet No.* 2 (*Fig.* 3)

Description	Flow Sheet	
	No 2	No. 1
Assay of mill feed	1.72	1.67
Assay of flotation machine feed	1.72	1.32
Assay of flotation machine tails	0.46	0.43
Assay of lower table tails	0.29	0.30
Assay of general concentrates	32 71	31.72
Recovery upper tables + flotation machine	......	75.30
Recovery flotation machine	74.30	
Additional recovery on lower tables	9 40	7.40
Total recovery	83.70	82.70
Tonnage per deck on lower tables	25.70	23.20
Tonnage per deck on upper tables	......	87.00
Oil consumption, cresol + turpentine in pounds per ton	0 95	0.82

REMARKS.—The recoveries are calculated from the feed and tailings assays and the assay of general concentrates.

CONCLUSIONS.—Roughing on upper tables reduces the tailings assay of the flotation machine slightly, while it does not seem to affect the assay of the lower tables appreciably. Increased recovery results from the use of the lower tables.

Table 1 gives the résumé, on the strength of which it was decided to use secondary tables only and to leave the preliminary table treatment (ahead of the flotation treatment) out of the flow sheet to be adopted in the concentrator under construction. That this conclusion is justified seems to be borne out by the following facts:

In January and February, 1914, the lower table floor was not in operation. For this reason, the flotation tailings were not treated on tables. There was, however, table treatment ahead of flotation, the following results being obtained: Table recovery, 39.42 per cent.; flotation recovery, 33.35 per cent.; total recovery, 72.77 per cent. From March to August, 1914, various flow sheets were tried; tables were sometimes used preceding and sometimes following flotation, and sometimes in both places. The figures on recovery are, for this reason, not comparable with the preceding ones. From September to December, 1914, no tables were used preceding flotation; the flotation tailings were, however, treated on tables. The results obtained were: Flotation recovery, 70.83 per cent.; table recovery, 7.51 per cent.; total recovery, 78.34 per cent.

Callow Flotation Installation

While the test mill was in operation, Mr. Callow, President of the General Engineering Co., advised the Inspiration company that he had invented and perfected a new flotation process which, in his opinion, would give as good, if not better, results than the machine of the Minerals Separation Co. As a consequence, arrangements were made to add a unit of machines of the Callow type. The flow sheet using Callow cells is illustrated in Fig. 4. It consisted of four rougher flotation cells, which served for the production of a low-grade concentrate, and another cell of the same construction which was supplied for the purpose of cleaning the concentrates made on the rougher cells. The tailings resulting from the cleaner cells were returned and mixed with the pulp entering the rougher cells. With the machine was furnished a Pachuca tank for the purpose of mixing flotation oil into the pulp entering the plant, should it be more desirable to do this in addition to or in place of feeding the oil to the grinding machines, as had proven useful both in the small and in the larger test plants. As a matter of fact, the use of the Pachuca tank was abandoned after several months' operation.

The Callow cells work on a different mechanical principle from that underlying the Standard Minerals Separation machine, inasmuch as no movable parts are used for the purpose of producing the fine dissemination of air with the pulp, which seems to be essential for any kind of flotation machine; while the Standard Minerals Separation machine depends on the operation of fast-revolving impellers, the dissemination is effected in the Callow machine by blowing air through a porous blanket, which forms the bottom of each cell.

The Callow machine, illustrated in another paper, is ingeniously simple. The feed enters at the upper end of the inclined bottom, while the tailings are discharged through the float valve at the lower end, the concentrates overflowing at the top of the flotation tanks. The porous bottom of the cell is set on an incline to make possible the treatment of feed containing coarse sand. The movement of such sand particles from the feed to the discharge end is thereby accelerated.

The mechanical principle of aeration and agitation by the admission of air through a porous medium, which has been termed "pneumatic flotation," was undoubtedly discovered by Mr. Callow[3] and his asso-

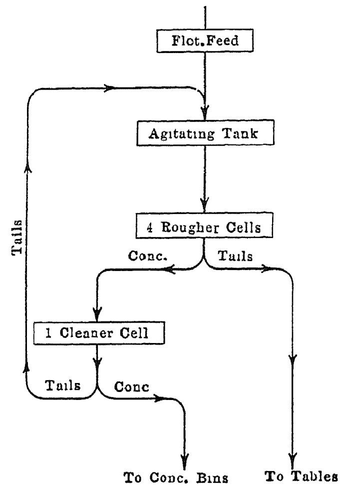


FIG. 4.—FLOW SHEET OF EXPERIMENTAL CALLOW FLOTATION PLANT.

ciates independently of other inventors, although an earlier patent had been taken out for the same thing in England by Minerals Separation (British Patent 10,929, May 3, 1910) and long before the installation of the Callow machine at this mill, Inspiration ore had been tested in New York by a pneumatic flotation machine constructed jointly by Messrs. Flinn and Towne.[4] The results of this test were so good that a small test plant of this system was installed at Inspiration.

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[3] Mr. Callow refers to his installation at the National Mill, Mullan, Idaho, April, 1914, as the first successful commercial plant utilizing the pneumatic principle.

[4] This test was made in the presence of Dr. L. D. Ricketts, Consulting Engineer, and W. D. Thornton, Vice-President of the Inspiration company. According to F. B. Flinn, a 600-ton plant of the Flinn-Towne flotation system was shipped to the Tezuitlan Copper Co. in Mexico in the spring of 1913. This could not be put into operation on account of the political conditions prevailing then. Otherwise, this would have been the first commercial pneumatic flotation plant. R. C. Canby informs me that some tests carried out by him suggested the idea to Messrs. Flinn and Towne of constructing a flotation machine to produce the desired fine dissemination of air by blowing air through a porous medium. [Editor's Note: See discussion by R. C. Canby, p. 645.]

Flinn-Towne Installation

The Flinn-Towne machine, as mentioned, utilizes the pneumatic principle also, but the application is somewhat different. An illustration of a single Flinn-Towne machine is given in Fig. 5, while Fig. 6 shows the outline of the installation. The cells are constructed in the

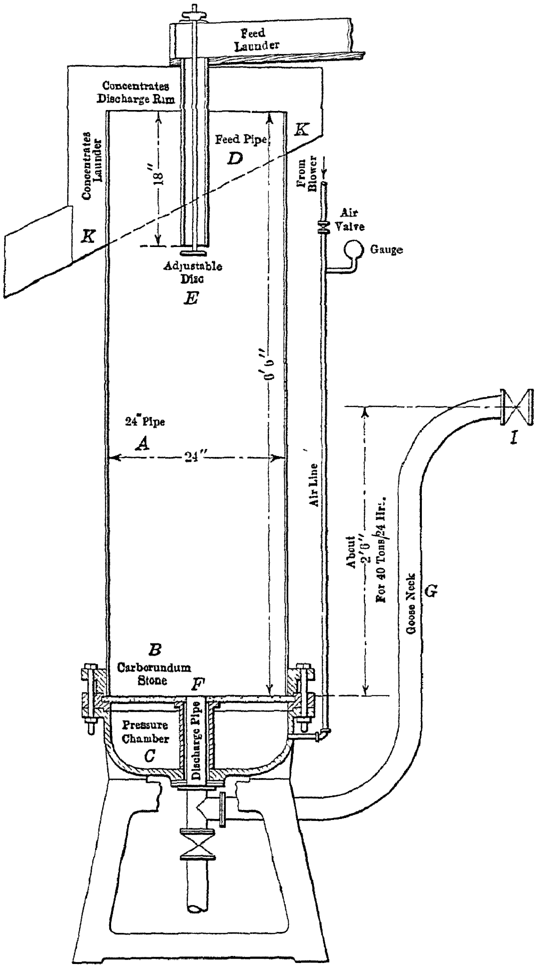

FIG. 5.—FLINN-TOWNE FLOTATION MACHINE.

shape of cylindrical tanks, the bottoms of which are formed by the porous medium. The feed enters near the top in the center of the cylinder, while the tailings leave the machine through a center hole in the porous medium. In the Flinn-Towne installation, only one roughing machine was provided, while additional cells served for the purpose of cleaning

both rougher tailings and rougher concentrates. The tailings produced by the concentrate cleaner cell and the concentrates from the tailings cleaner cell were returned to the head of the roughing cell. In place of the canvas blanket of the Callow machine, Messrs. Flinn and Towne in the demonstration test at Inspiration used carborundum stones as the porous medium through which the air is injected into the pulp.

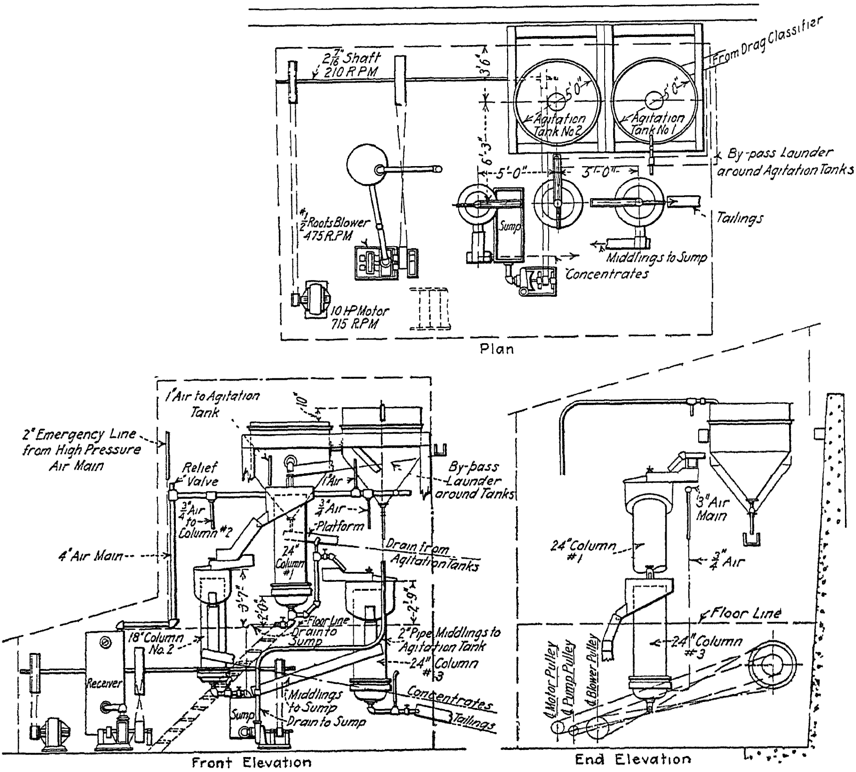

FIG. 6.—OUTLINE OF FLINN-TOWNE FLOTATION INSTALLATION.

Cole-Bergman Installation

The Flinn-Towne machine was withdrawn from the contest after a competitive test between this machine and others had run for several months, although the results obtained looked very encouraging. Its place was taken by a flotation machine designed by Messrs. Cole and Bergman.

Their machine is illustrated in Fig. 7. In principle the cells are similar to those of the Flinn-Towne machine. Evidently, the designers tried to improve on this system by mechanically developing the idea underlying the machine, and by changing one point, which in their opinion formed a weak part of the Flinn-Towne machine. This is the

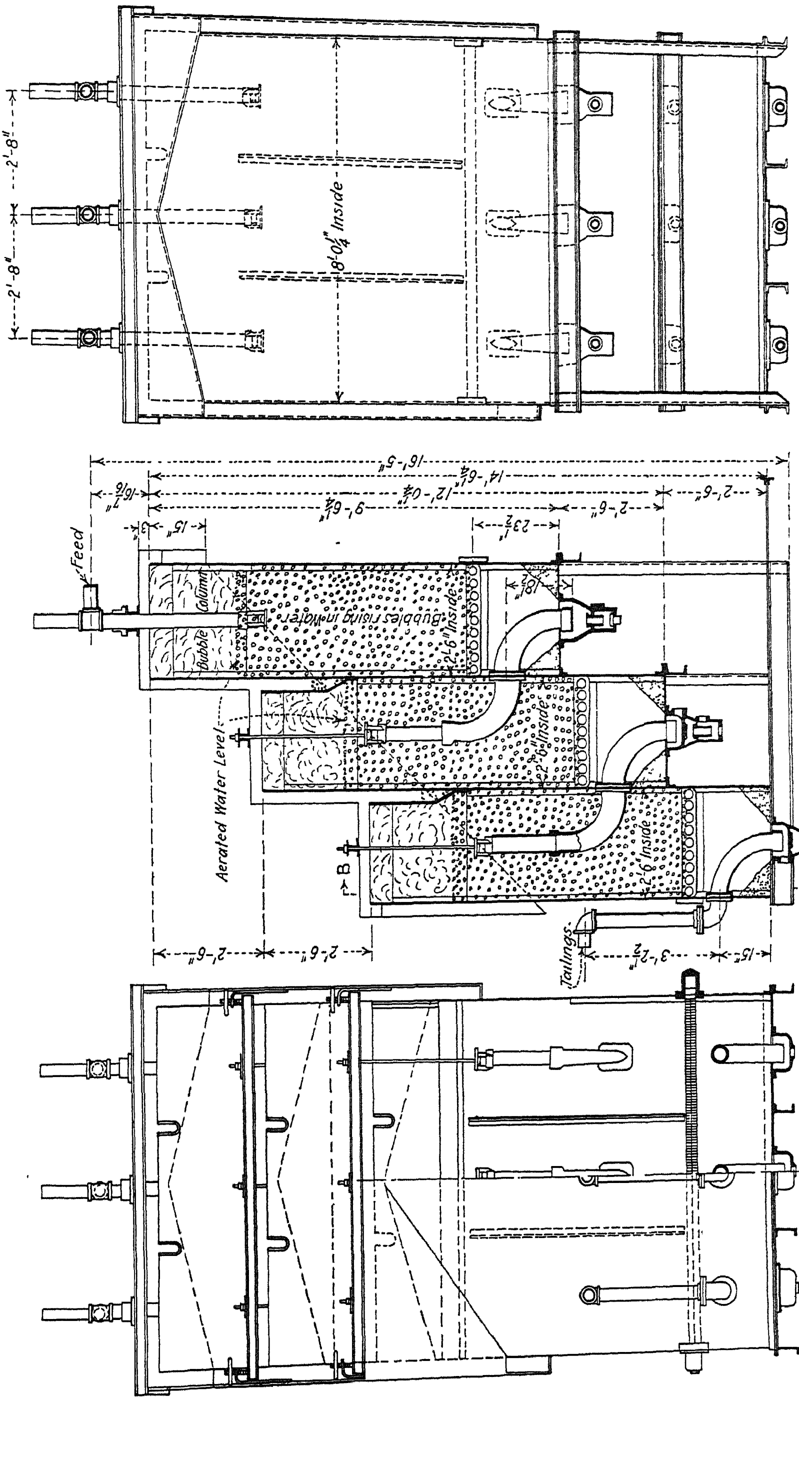

FIG. 7.—COLE-BERGMAN FROTHING CLASSIFIER.

construction of the porous diaphragm, which, as explained below, was formed by a round carborundum disk. While carrying on the test of the Flinn-Towne machine, it proved necessary occasionally to wash the carborundum stones by injecting a water pipe from the top and even by removing the stones and cleaning them with water, acids, etc. Messrs. Cole and Bergman ascribed this deficiency to the fact that on account of the horizontal position of the porous medium, sand had a tendency to lodge thereon and to impede or prevent the passage of air through the pores. They substituted, therefore, a system of perforated tubes covered with a suitable fabric, such as canvas or flannel. This led to the construction of a porous medium in the form of a grate, as shown in the illustrations. Their idea seems to be a very happy one, as it makes it possible to apply the flotation process to ore pulps containing comparatively coarse particles of sand; for instance, it seems possible to treat with this machine ore mixtures that contain sand particles as coarse as 10-mesh and perhaps even coarser. Using this machine, the millman is now in a position to treat slime by flotation without the necessity of removing it from admixture with sand. The Cole-Bergman machine has given good results in our tests, as have the rest of the machines utilizing the pneumatic principle. It was operated in conjunction with a smaller machine of the same type installed for the purpose of cleaning the concentrates produced on the larger machine.

Methods Followed in Competitive Tests

In the interest of a fair contest between the flotation machines of the different types, each machine was provided, as nearly as possible, with the same character of pulp as the rest. In the beginning this was accomplished by providing each machine with one or more pebble mills to which feed was sent in such quantities that a mill product of practically the same fineness resulted in each case. When the results of the competitive tests began to approach each other very closely, the equalization of the pulp furnished to the different machines was still improved upon by mixing the ground product discharged from all the mills, and sending it to a divider which permitted the division into as many parts as there were competing machines, and in any proportion desired. As it had been decided in former experiments not to install tables ahead of the flotation process, no preliminary table treatment was used during the competitive tests, but each flotation system was furnished with a set of tables for the retreatment of the flotation tailings. It developed that this was essential in trying to arrive at a fair valuation of the machines, as some of the machines produced tailings from which additional mineral could be extracted by the table treatment more easily than from the tailings of other flotation machines, the cause evidently being that

flotation machines of one type have a tendency to save more of the finer particles, while flotation machines of another type permit of a better recovery of coarser grains. Those flotation machines that make a good recovery on the sand, but leave a larger percentage of mineral in the slime, will not benefit from a subsequent table treatment as much as those of the other kind.

Advantage of Pneumatic Flotation Machines

In the course of the tests at Inspiration, the fact became apparent that, by the application of the pneumatic principle, a higher recovery of the slime is made possible than without the utilization of injected air, although the Standard Minerals Separation machine also gives excellent results when the tonnage is reduced to a considerably lower figure than the machine is supposed to be able to treat economically.

Therefore, in cases where high power consumption is of little importance, the Standard Minerals Separation machine will fill the requirements of a flotation machine remarkably well. When, however, the power consumed has to be seriously considered, it seems advantageous in the light of the Inspiration experiments to make use of pneumatic flotation for the reason pointed out above, *i.e.*, that poor work of flotation machines on sands can be made up by a subsequent table treatment, while poor work on slime cannot. Within certain limits, it is more important to insist on machines effecting a good slime recovery than on machines making a good sand recovery.

Pneumatic Machines of Minerals Separation Co.

The Minerals Separation Co., realizing this, offered to put in their subaeration type of machine which added the advantages resulting from the use of injected air to the advantages which their Standard machines seemed to have in saving coarser mineral. This resulted in the construction and testing successively of two of their subaeration type machines, one in which the old spitzkastens still were retained, and another one in which they were dispensed with. As seen from the illustration, Fig. 8, provisions were made in the former machine to admit air to the pulp conduits carrying the pulp from the spitzkastens to the agitating compartments next in succession, in order to make this air available to lift mineral to the top of the spitzkastens where it can be skimmed off. The agitating compartments are closed at the top by covers set on an incline in such a way that the air rises to the top in the agitating compartments and is conducted to the spitzkasten, being thereby utilized for the purpose of carrying the mineral values.

This construction of the Minerals Separation Co. was, however, only

an intermediate step toward the design of the second-named machine, a machine of greater simplicity, which is similar in principle to the one

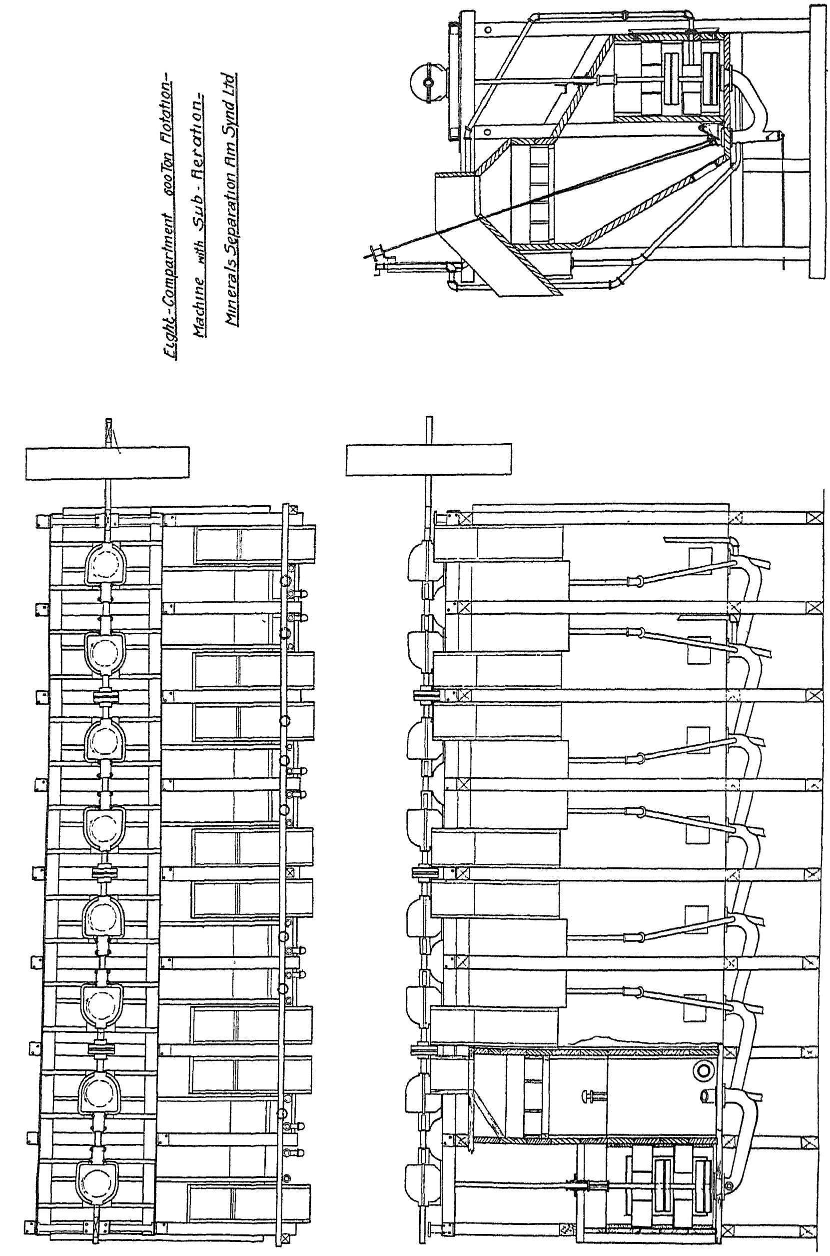

Fig. 8.—Minerals Separation Subaeration Type of Flotation Machine.

illustrated in Fig. 9. The machine consists of a long rectangular tank without any partitions, in which a number of agitators revolve, while air is

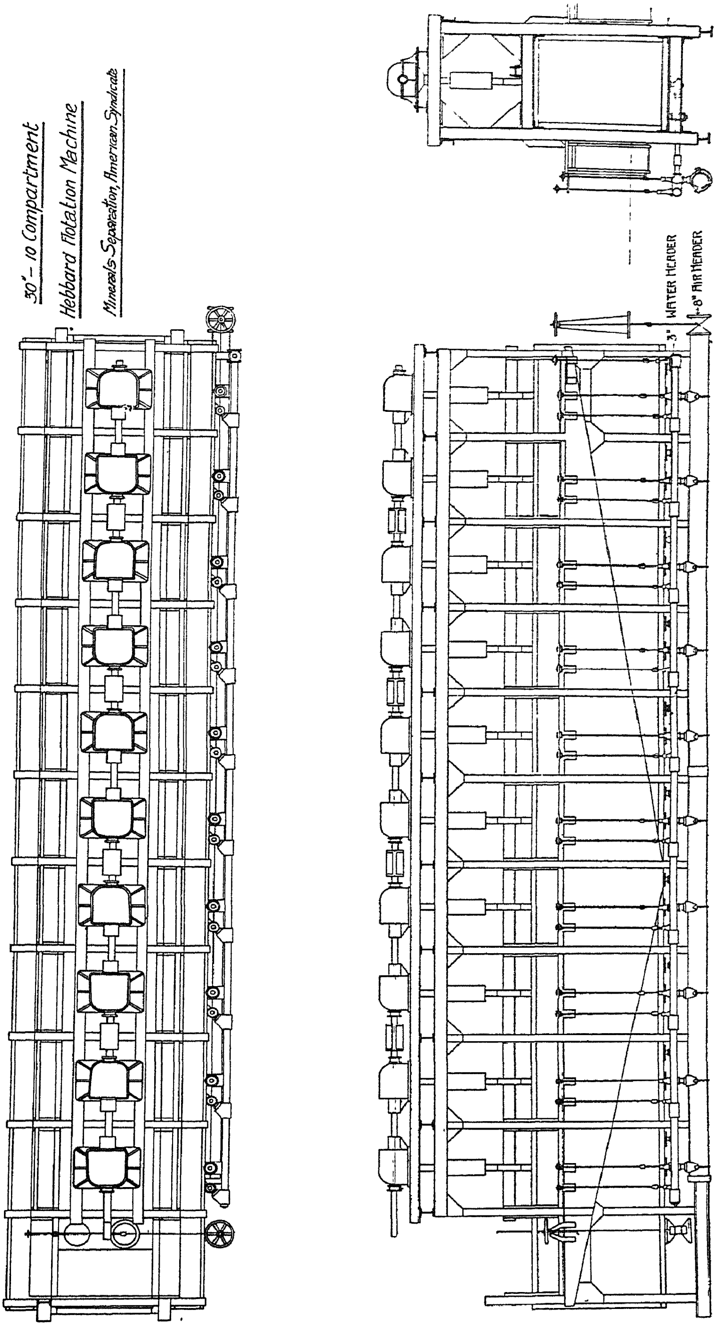

FIG. 9.—HEBBARD TYPE OF MINERALS SEPARATION MACHINE.

injected underneath each impeller. We call the machine the "Hebbard Type" Minerals Separation Machine.

In order to limit the agitation to the lower part of the machine and to provide an area of comparative quietness in the upper part, a system of baffles is arranged above the agitators. As a consequence, the froth is given a chance to separate out in the upper portion of the rectangular tank, and flows off on both sides lengthwise into launders provided for the purpose.

This machine marks an important progress in the development of the Minerals Separation flotation systems toward increased recovery of copper, at least as far as the Inspiration ores are concerned. In our tests it effected a higher recovery of the fine mineral particles than the old machine had accomplished. The machine also has the advantage of greater simplicity and relatively low power consumption. The only drawback that we have found in the operation of the machine, extending over a number of months, is that the agitating shafts, which are suspended from above, have a tendency to bend and, if not straightened, soon cause the impeller blades to strike against the baffles, breaking one or the other.[5]

Float Skimming Device

While the competitive tests of the different flotation machines were being conducted, it was observed that large quantities of mineral froth frequently appeared in the tailings launders. The idea suggested itself to remove the froth, thus increasing the recovery obtained in the flotation machines and concentrating tables by an additional small amount. After some experimenting, a way was found in which this could be easily accomplished. The tailings launder was widened and deepened for some distance. The tailings stream was forced to pass underneath a baffle on entering the widened space, and was also forced to travel under another baffle board at the end of the space, thus being made to rise through the restricted area formed by the tailings end baffle and the back-wall of the float-saving device, which is illustrated in Fig. 10.

Contrary to what might be expected, tailings sand of the fineness of Inspiration tailings will not lodge in the widened-out space unless the length is greater than a certain critical distance, nor will it choke the upward channel in the tailings end of the machine. When conditions demanded the construction of the float-saving device in greater length, it was found that the accumulation of sand, which has a tendency to form in the middle of the machine, can be avoided by arranging an additional baffle not reaching quite to the bottom in the center of the machine. If

[5] This trouble is alleviated in the Hebbard machines turned out more recently by the Minerals Separation Co. by the provision of suitable bearings for the impeller shafts in the bottoms of the machines.

the machine is built longer yet, more than one intermediate baffle is necessary. The explanation for this behavior of the pulp, which at first glance might seem paradoxical, is that, on account of reduced passageway between the lower part of the baffles and the bottom of the machine, the pulp is forced to travel through at a speed so high that it will not permit the settling out of sand. As a consequence, a hydrostatic head will establish itself between front and back of such a baffle and has to be taken care of in the design. It is found, however in actual practice, that the

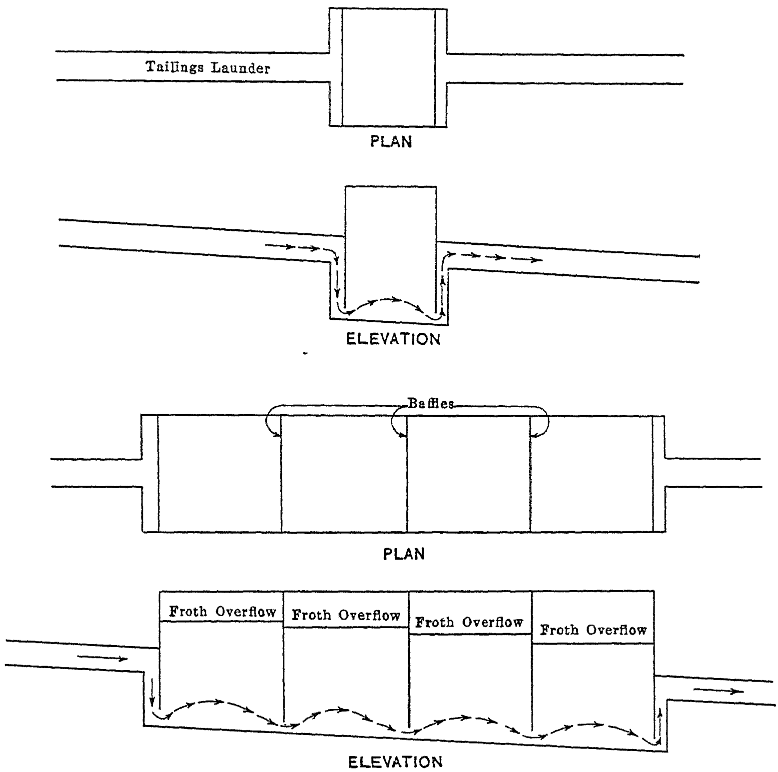

Fig. 10.—Diagram Showing Origin and Development of Inspiration Flotation Machine.

hydrostatic head required to keep the sand from settling out by creating an accelerated current underneath the baffle is relatively small.

A similar consideration explains the fact that the upward tailings passage at the end of the machine has very little, if any, tendency to choke; to take care, however, of possible choke-ups that might be caused by the discharge of coarse rocks, pieces of wood, etc., into the pulp, it was found advisable to arrange some air jets in the bottom of the upward tailings passage to assist in clearing it whenever necessary.

Experimental Inspiration Flotation Machine

This device was in actual operation for several months at our test mill and made itself pay well by adding about 1 per cent. to the recovery

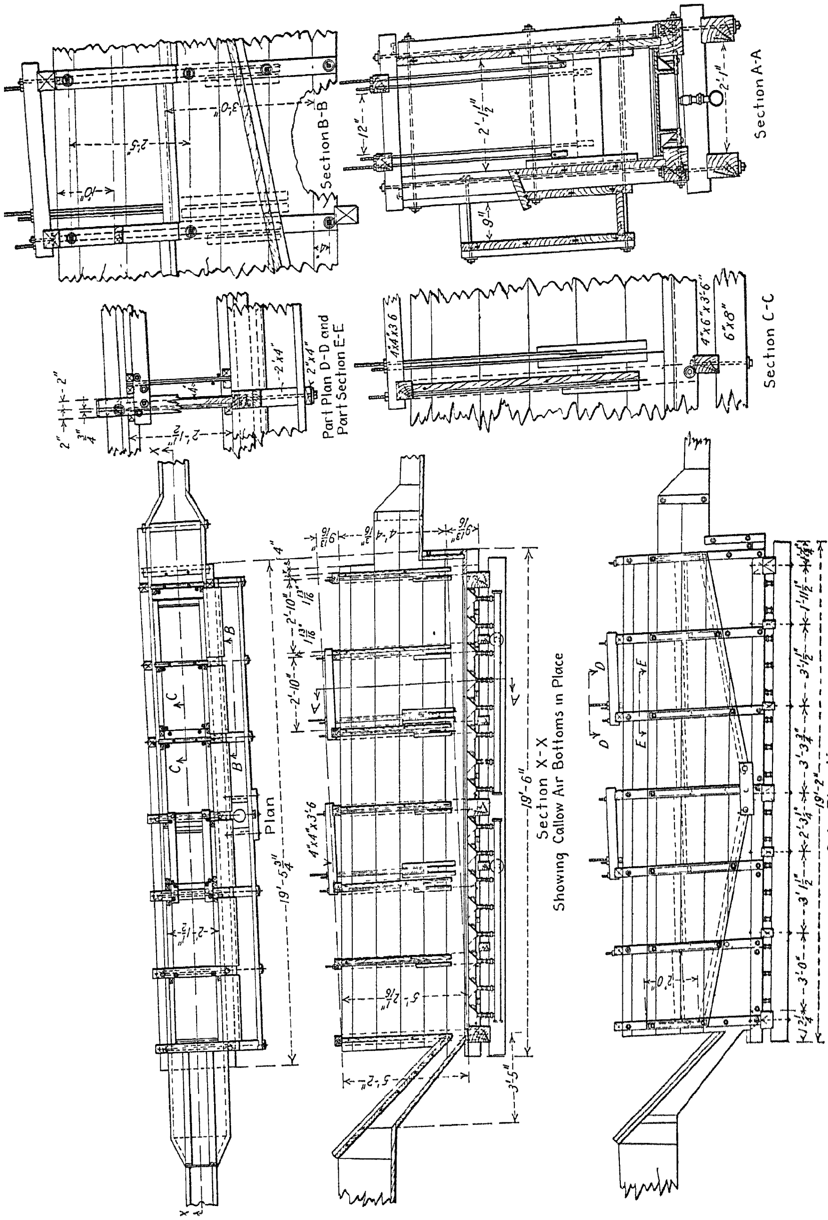

Fig. 11.—Experimental Inspiration Flotation Machine.

actually obtained. The question then presented itself as to whether this device, which was successful in saving mineral escaping from the flotation

machines could not be transformed into a flotation machine itself by arranging for the injection of finely distributed air into the pulp passing through the machine. The development of the idea led to what we call our Experimental Inspiration Flotation Machine, which was built on a small scale, that is of a size to treat up to 100 tons in 24 hr., and was placed in competition with the other flotation machines already in existence. This machine is illustrated in Fig. 11. The air was injected through a porous bottom arranged in a way similar to the bottoms of the Callow and Flinn-Towne machines (as a matter of fact, a Callow bottom was used in the first tests) and the concentrate flowed over the edge on

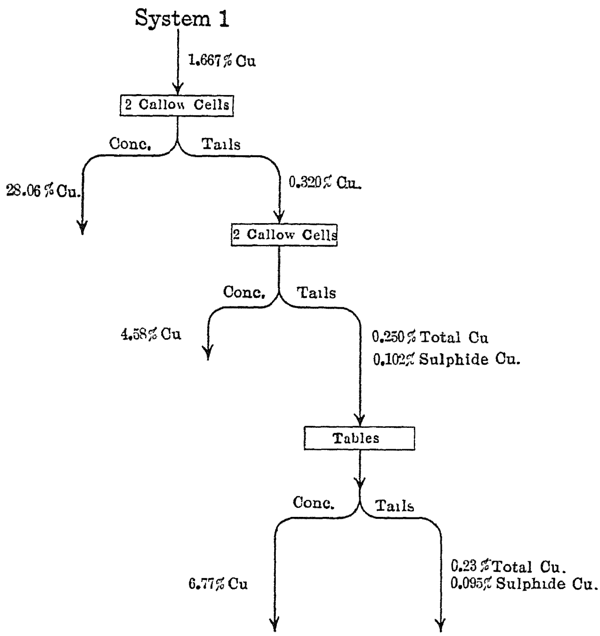

FIG. 12.—TESTS WITH CALLOW CELLS IN SERIES.

one side of the flotation tank. The mode of operation followed was also similar to the one developed for the other air machines. A low-grade concentrate was produced and retreated on a similar machine of the same construction, called the cleaner machine, the tailings from which were returned to the head of the roughing flotation machine. As will be seen from the drawing, the construction of this machine is extremely simple and comes pretty near to an ideal of Mr. Mills, our general manager, who prophesied that the flotation machine of the future would be nothing but a launder with provisions for injection of air. The method of effecting the concentration in two stages was also followed when the new types of the Minerals Separation machines were put into commission.

Arrangement of Callow Cells in Series

In principle, the Inspiration machine shows one difference from the Callow machine; viz., that instead of splitting the pulp between a great number of machines, it is forced to travel through a series of compartments in succession. This is the same policy that had also been followed in the construction of the Minerals Separation machines. The question then came up, whether it might not be of advantage to arrange the Callow cells also in series. To test out the idea, the flow sheet of the Callow plant was changed (see Fig. 12). The primary feed was sent to two of the cells, the other cells receiving the tailings from these primary cells.

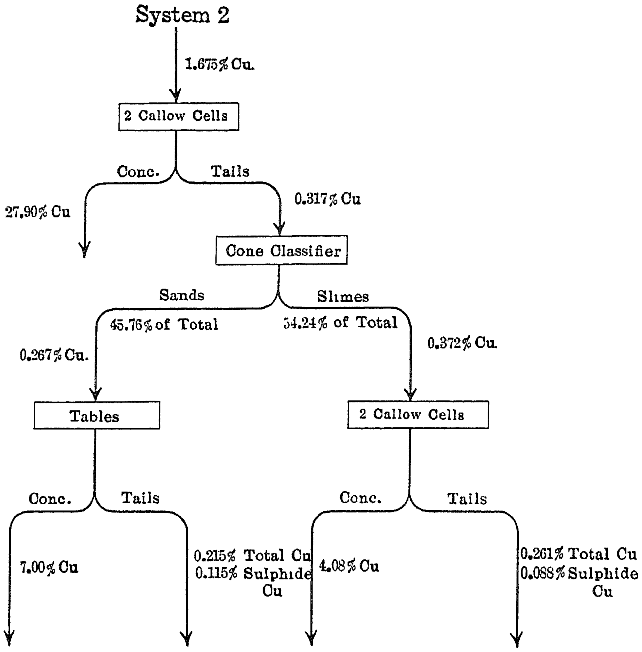

FIG. 13.—TESTS WITH CALLOW CELLS IN SERIES WITH CLASSIFIERS BETWEEN.

This arrangement showed at least no disadvantage over the multiple treatment; it seemed superior to the other system in the one respect that it showed at a glance whether the cells were operated well, because in that case most of the concentrate would be produced on the primary cells while the secondary cells would produce a rather light-colored froth and would add only a small fraction to the recovery made on the primary cells. Both primary and secondary concentrates were treated jointly in a cleaner cell, while the tailings from the secondary cells were conveyed to a number of concentrating tables for retreatment.

A variation of the latter flow sheet was also tested (compare Fig. 13).

Between the primary and secondary cells, a cone classifier was interposed for the purpose of making a separation between the sand and slime. The slime was sent to the secondary flotation machines, while the sand was passed to tables. This latter flow sheet was finally decided upon for the Callow installation in the large concentrator. Tests made with the object of establishing which of the two is superior, resulted in a tie. Some of the figures on this point are given in Table 2.

TABLE 2.—*Retreatment of Tailings from Two Callow Cells. Comparison of Results Obtained from Two Different Methods as Shown in Figs.* 12 *and* 13. *Screen Analysis of Feed and Tails of Secondary Callow Cells*

Mesh	System 1									System 2							
	Feed				Tails					Feed				Tails			
	Per Cent Weight		Per Cent. Cu	Cu Contents	Per Cent Weight		Per Cent. Cu	Cu Contents	Per Cent. Oxide Cu	Per Cent. Weight		Per Cent. Cu	Cu Contents	Per Cent. Weight		Per Cent Cu	Cu Contents
	Cum.	Indiv.			Cum.	Indiv				Cum.	Indiv			Cum	Indiv.		
65	5.0	5.0	0.29	0.014	5.0	5.0	0.21	0.011	0.05								
100	19 0	14.0	0.25	0.035	19.2	14.2	0.15	0.021	0.06	3.8	3.8	0.20	0.007	4.0	4.0	0.17	0.007
150	32 2	13.2	0 29	0.039	32.4	13.2	0.16	0.021	0.10	10.6	6.8	0.25	0.017	10.8	6 8	0 14	0.010
200	42 4	10 2	0.33	0.034	42 8	10.4	0.21	0.022	0.11	18.6	8.0	0.30	0.024	19.0	8 2	0.25	0.021
	100 0	57 6	0.35	0.201	100.0	57.2	0.27	0.155	0.20	100.0	81.4	0.40	0.326	100.0	81.0	0 30	0 242
Assay calc . ..			0 32	0 323	.....	....	0.23	0.230	0.15	.....	....	0.374	.....	.....	...	0 28	0 280
Assay direct .			0.32	.....	.....	....	0.23	.....	0.15	.....	....	0.370	..	....	....	0.26	... .

NOTE.—The concentrates from the flotation cells were in both cases retreated in half-size Callow cells, the retreatment tailings being returned to the head of the primary cells.

	Ton per Primary Cell
Tonnage rate in system 1....	49.3
Tonnage rate in system 2..........................	47.5

CONCLUSIONS.—No advantage is apparent in favor of either system. System to be installed should be decided from mechanical points.

Variations of Flow Sheet

In the way of other flow sheet variations, etc., only one thing was tried to any extent; that is, whether it is wise to send the cleaner tailings back to the head of the rougher machine for retreatment. In the limited time at our disposal, it proved impossible to substitute something better, but the chances are that the same holds true of flotation that is true for water concentration; viz., that middlings are sent back for retreatment mainly because the designer does not know anything better to do with them. In principle it seems wrong to follow this practice, because, when heavy copper losses occur in concentrating machines which express themselves in high tailings, there is always a doubt as to whether this is due to poor

recovery made in the primary treatment or to heavy losses incurred by middlings sent back for retreatment.

Coal Tar as Flotation Agent

It was to be expected, when the flotation process was installed in our test plant, that there would be ups and downs in the recovery because the process was rather new, especially in its application to chalcocite ores. For the month of June, 1914, the recovery obtained in the test mill showed a sudden drop, and the serious problem confronted us of establishing the cause and finding a remedy. Some of the flotation experts suggested that it might be due to the fact that a new shipment of cresylic acid which had been received and used at that time might not fill the specifications of being 98 per cent. pure. We did not feel competent to say whether the impurities actually amounted to more than 2 per cent. We were, however, inclined to think that perhaps cresylic acid, which is, as is well known, one of the products resulting from fractional distillation of coal tar, might not represent the fraction most suitable for the flotation of our ores. Having no coal tar available in the neighborhood, we proceeded to make some by distilling a sample of ordinary New Mexico soft coal and separating the tar thus formed into the fractions distilling off at different temperatures.

Our facilities for testing oils were limited at that time. The Minerals Separation Co.'s representative did not believe in small-scale tests, and for this reason did not recommend experiments with small testing machines. Nevertheless, it seemed desirable to have something with which to carry out small-scale laboratory experiments. Dr. Ricketts, who was aware of our troubles and realized the importance of such tests, was kind enough to send us a little electrically operated emulsifying machine, which served admirably for qualitative tests. We also built a testing machine based on the principle of the Standard Minerals Separation machine, with the difference, however, that instead of sending the pulp from one agitating compartment to a spitzkasten and then into another agitating compartment and spitzkasten, we made the pulp return from the first spitzkasten to the original agitator, forcing it to revolve in a closed circuit. Our laboratory machine is illustrated in Fig. 14. Lately, a machine based on the same principle has been put on the market and is sold by the Denver Fire-Clay Co. Thus, we had a chance to try the different fractions of our home-made coal tar.

The chemist who conducted these tests (Mueller) hit on the idea that it might be well, in addition to trying the different fractions, also to test the coal tar as a whole. The results were very surprising, since they showed that by the addition of crude coal tar we could effect a greater recovery than we were able to obtain by the use of highly refined cresylic

. From this point dates our experience that it is better to use coal than soluble flotation agents like cresylic acid to save coarse mineral. ;ylic acid is an extraordinarily good agent for producing froth, but

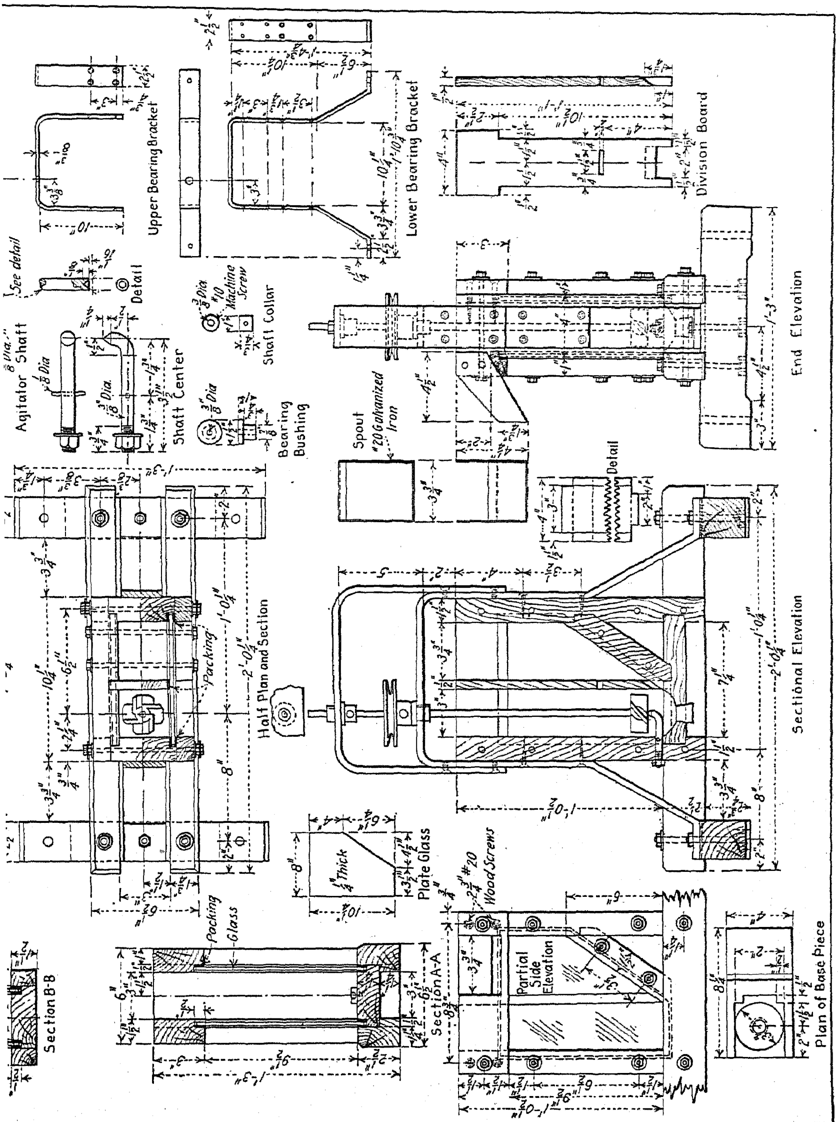

FIG. 14.—LABORATORY FLOTATION MACHINE.

froth which it produces does not seem to have as much carrying power coarse mineral as that produced by coal tar. Not all coal tars are ıally good for this purpose. Tests in laboratory machines easily show difference between coal tars of different origin.

The system that we used to carry out such tests has been described, although without authorization, by William A. Mueller, one of my former assistants who took part in these laboratory tests.[6]

It is difficult to utilize coal tar in plants using flotation supplementary to gravity concentration, on account of the fact that it is not easy to effect a good amalgamation of tar with the pulp in agitating tanks, and even in mechanical flotation machines. The use of coal tar lends itself very well indeed to the system of feeding tar into the grinding machines, a system that, as mentioned above, had been worked out in our small test mill and patented by Mr. Chapman.

The company is indebted to Mr. Callow for proving the merits of coal-tar creosote as a flotation agent by using it in his demonstration plant at Inspiration. After we had established the value of coal tar by laboratory tests, and while efforts were being made to obtain it commercially, he applied creosote successfully. We have continued to use it for a long time, mostly in combination with coal tar, and have only recently dropped it, as we find crude coal tar cheaper and better.

Experience with Primary Slime

After the first difficulty that we encountered in our large-scale tests had been solved by the introduction of coal tar into the flotation-oil mixture so far used, things went along fairly well for some time until a new difficulty was encountered. It happened that once in a while an abundant froth was produced on the flotation machines, but this froth seemed to have very little carrying power for the mineral contents of the ore and held mainly finely divided gangue. It was observed that this phenomenon occurred with special severity whenever the ores shipped to the test mill contained a large amount of fines and a small amount of coarse rock. It was attributed, therefore, to the presence of what may be called primary slime, *i.e.*, slime not formed by the crushing of ore in the mill, but originating from the mine. That the falling down of the flotation machines was caused by variation of the ores was proven by the fact that samples of the refractory ores when treated in the laboratory testing machine gave as unsatisfactory results as the corresponding ore did in the big flotation machines. To demonstrate that the presence of the original slime caused the trouble, samples of the mill feed were separated by screening them on a 200-mesh screen. The oversize, when reduced to the proper fineness for flotation treatment, did not offer the least trouble and yielded a high-grade concentrate in the laboratory machines, while the undersize proved extremely refractory (refer to Table 3). The same experiment was repeated on a large scale,

[6] William A. Mueller: Use of Coal Tar in Flotation, *Engineering and Mining Journal*, vol. 100, No. 15, p. 591 (Oct. 9, 1915).

the products from a Marcy ball mill crushed to about 6-mesh being treated on a drag classifier. The oversize, after being reduced in pebble mills to the necessary fineness for flotation treatment, was sent to one group of flotation machines, while the overflow from the drag classifier was sent to another group. Fig. 15 illustrates the results of this experiment. While the reground oversize yielded remarkable lean tailings,

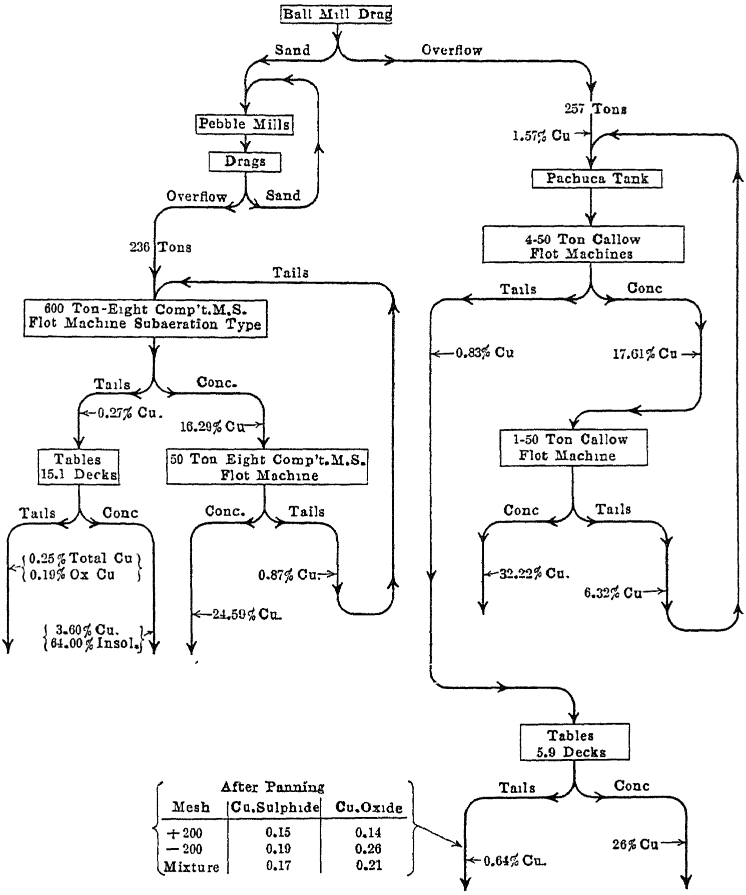

FIG. 15.—TESTS TO DETERMINE EFFECT OF SEPARATE TREATMENT OF SLIME AND REGROUND SAND.

the product containing the primary slime could not be treated advantageously by flotation. We were almost on the point of concluding that in order to get the best results, a separation of the primary slime should be made in our large mill and flotation should not be entirely relied upon for the treatment of the slime. Before finally deciding on this point, however, some additional laboratory experiments were made to study the influence of the primary slime on flotation. These tests established

TABLE 3.—*Screen Analysis of Ore Drawn from Mill Bins and Flotation Tests of Different Screen Sizes*

Screen Size	Weights, Per Cent		Per Cent Total Copper	Copper Contents	Per Cent Oxidized Copper	Contents Ox Cu	Flotation Results Per cent Total Copper		Recovery Per Cent
	Cumul.	Indiv.					Tails	Midds.+ Conc.	
Inches									
+ 1	30.8	30.8	1.05	0.324	0.13	0.40	0.22	12.8	80.5
+ ½	47.9	17.1	1.34	0.229	0.11	0.19	0.22	17.4	84 7
Mesh									
+ 4	70 9	23.0	1.44	0.332	0.12	0.28	0.17	17.1	89.0
+ 8	75.2	4 3	1 42	0.061	0.12	0 05	0 17	16.3	88.8
+ 14	80.3	5.1	1.77	0.068	0.13	0.07	0.26	16.8	86.7
+ 28	84.4	4.1	2.14	0.080	0.14	0.06	0.61	16.3	74.3
+ 48	87.5	3.1	2.62	0 081	0.19	0 06	0 72	11.3	77.5
+100	89 9	2.4	3.29	0.079	0.24	0.06	0.63	22.5	83.1
+200	91.9	2 0	3.16	0.063	0.27	0.05	0 60	20.1	83.5
−200	100.0	8.1	1.53	0.124	0.43	0 35	1 57	1.48	None
	.. ..	100.0	1.44	1.441	0.16	0.157			

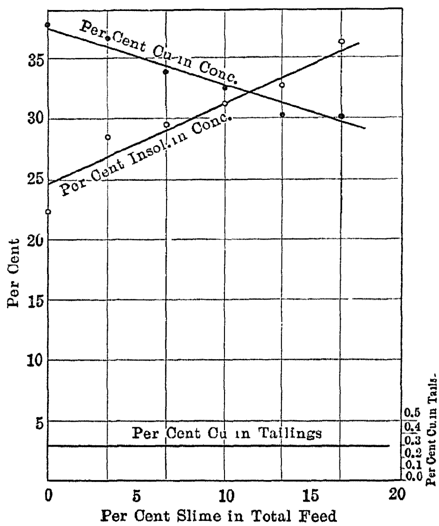

FIG. 16.—INFLUENCE ON FLOTATION OF MIXING PRIMARY SLIME AND REGROUND SAND.

The + 1-in. and + ½-in sizes resulting from the screen analysis were ground to 10 mesh, mixed with varying amounts of − 200 material and the mixtures ground with the usual amounts of oil in the laboratory ball mills. After this preliminary treatment, the samples were treated in the laboratory flotation machines.

one fact quite well, which is, that when refractory slime is mixed with a sufficient quantity of coarse ore ground to the necessary fineness, in other words, when the percentage of primary slime in the flotation feed is kept

low, the slime loses its refractory character. I do not mean to say that by reducing the quantity of primary slime, the deleterious influence is reduced in proportion so that it cannot be detected as easily, but conclude from our tests that when the flotation feed contains a sufficient percentage of comminuted coarse rock, the primary slime contained in it can be treated more advantageously than it could by itself. Some tests bearing on this point are illustrated in Fig. 16. They indicate that as high an amount as 20 per cent. primary slime may be mixed with ground coarse ore without causing increased tailing losses.

The concentrates, according to this set of tests, seem to have a tendency, however, to carry more insoluble matter with an increasing percentage of slime.

Influence of Iron on Flotation

While these tests were in progress, we made another accidental discovery which proved very helpful to us. In our tests on the most economical way of reducing the ore to the fineness necessary for flotation we had, among other machines, a ball mill in competition with pebble mills. In the ball mill, steel balls performed the duty that in pebble mills was done by flint pebbles.

For a while, the ball-mill discharge was treated on one flotation machine, while the pebble-mill discharge was treated on a group of others. While this flow sheet was being followed, we thought we noted that a flotation machine treating the ball-mill product showed the influence of the primary slime to a less extent than the flotation machine treating the pebble-mill product. In a discussion with Dr. Ricketts and Mr. Mills, the question was raised as to whether the iron introduced in the mill pulp by the attrition of the balls might not have something to do with the fact. The question was accordingly made the subject of some laboratory experiments. The results of a series of such experiments are represented in Table 4, and proved conclusively that the iron had a beneficial influence on flotation in counteracting the harmful effect of the primary slime. This discovery was one of the inducements for installing ball mills in the big concentrator, while originally pebble mills had been considered for this purpose.

We have not yet reached a point where we can safely give the reason for the action of the iron introduced into the flotation pulp. It is sure, from the experiments referred to, that the same results, as by grinding with balls, could be obtained by introducing the iron in finely divided form, say in the form of filings, into a pebble-mill pulp. We supposed for a while, and I am not yet certain that this supposition is incorrect, that the metallic iron might react on the impurities contained in solution in the mill water and introduced therein with the primary slime. We find, as a matter of fact, that our ores contain very little in the nature of

TABLE 4.—*Effects of Iron and Other Solids on the Flotation of Refractory Copper Ores*

Test No	Grams Ore	Per Cent. Copper	Grams Copper	Concentrates			Recovery Per Cent.	Remarks
				Grams	Per Cent. Cu	Grams Cu		
F21	750	2 01	15 07	45	23.66	10 65	70.7	Added 10 g iron filings
F22	750	2 01	15 07	47	24.52	11.52	76 4	Added 10 g. iron filings.
F27	750	2.01	15.07	43	27.10	11.65	77.5	Added 2 g iron filings.
F28	750	2.01	15 07	47	26.90	12.64	84.0	Added 2 g. iron filings.
F45	750	2 01	15.07	51	23.84	12.16	80.7	Added 10 g. iron filings.
F46	750	2.01	15.07	48	25.60	12.29	81.5	Added 10 g. iron filings.
F47	750	2.01	15 07	27	8.80	2.38	15.8	Blank with no solids added.
F48	750	2 01	15 07	28	8.34	2 34	15.5	Blank with no solids added.
F49	750	2 01	15.07	62	20.14	12.49	83.0	Added 10 g. miscellaneous iron filings from shops.
F50	750	2.01	15.07	63	20.10	12.66	84.0	Added 10 g. miscellaneous iron filings from shops.
F51	750	2 01	15.07	60	18.82	11 29	75.0	Same as F49 and F50 by different observer.
F52	750	2 01	15 07	62	19.54	12 11	80 5	Same as F49 and F50 by different observer.
F53	750	2 01	15 07	29	6 16	1 79	11.9	Blank with no solids added.
F54	750	2 01	15.07	30	7 96	2.39	15.9	Blank with no solids added.
F55	750	2 01	15 07	59	21 08	12 44	82.7	Added 10 g. iron filings.
F56	750	2.01	15.07	56	25 80	14.45	96 0	Added 10 g. iron filings.
F64	750	2.01	15 07	65	19.52	12.69	81.9	Ground in mill with steel balls instead of pebbles.
L27	750	1 70	12.75	37	28 42	10.51	81.7	Blank on good flotation ore
L28	750	1 70	12.75	34	28 46	9.85	78 3	Blank on good flotation ore.
L29	750	1 70	12.75	76	5.66	4.30	33.4	Identical conditions as L27 and L28 but added 10 g. zinc filings.
L30	750	1.70	12 75	83	5 06	4.20	32 9	Identical conditions as L27 and L28 but added 10 g zinc filings.

soluble salts, and that whatever they do contain is mainly confined to the primary slime. For this reason, in laboratory tests we have tried repeatedly to substitute pure water for the water contained in the mill pulp. In every case, we have noted some improvements in the results obtained. We have also found that when we separate the water from the refractory pulp, treat it with iron filings, and add it to the original pulp again, we get a certain improvement in the recovery, but we have not been able to get an improvement equally as good as that obtained by direct introduction of finely divided iron into the pulp. For this reason, we have often thought that the effect of iron is physical rather than chemical in character. The iron exists in the pulp, at least partly, in the metallic form, as can be proven by the use of the magnet. If necessary, the effect of the iron could be increased by removing the iron contained in the tailings pulp by means of electromagnets and returning it to the mills or the flotation machines. Although we have considered the possibility of applying this fact commercially, we have not yet done so.

I have been told by several people that they have tested the influence

of finely divided iron on their ores and have not obtained any improvement whatever. This shows, evidently, that metallic iron is not a universal remedy for all flotation troubles, but as far as our primary slime is concerned, our experience leaves no doubt about its usefulness, and I believe the figures quoted above are positive enough to bear out my statement. As a curiosity I have included in Table 4 some figures showing the influence of the addition of zinc filings to ore that without the addition does not offer any difficulty in the flotation treatment.

Influence of Chemicals on Primary Slime

We also found that the harmful influence of primary slime can be counteracted by the introduction of certain chemicals; for instance, we

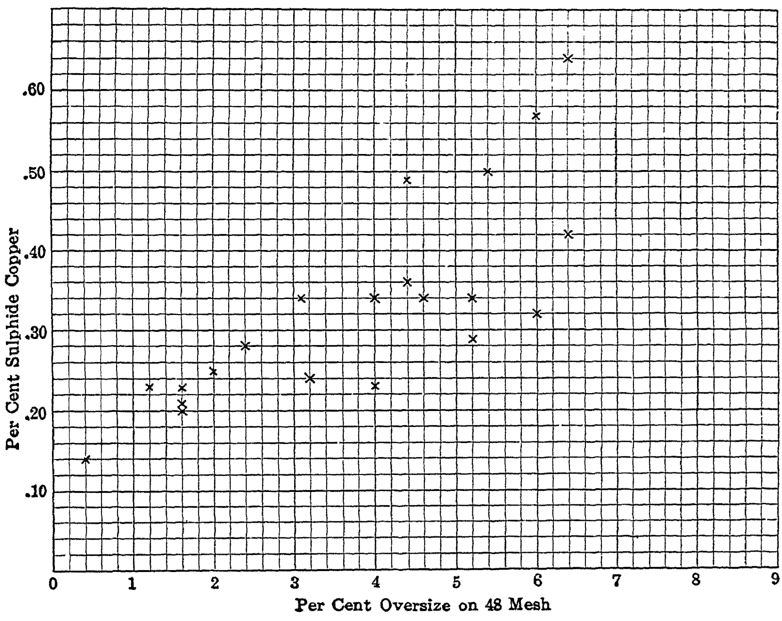

FIG. 17.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW FLOTATION TAILING.

have used advantageously sodium hydroxide or potassium cyanide. The use of sodium hydroxide was recommended to us by the Minerals Separation Co., which, I understand, applied it first in the plant of the Caucasus Copper Co.

Fineness of Crushing Desired

Tests to determine the most advantageous fineness for treatment of the ore according to our flow sheet, *i.e.*, flotation machines followed by

tables, naturally formed an essential part of our tests. Results of this character are plotted in Figs. 17 and 18 and seem to indicate that to get the best results grinding should be carried to such a point that not more than about 1 or 2 per cent. of the pulp will remain on a 48-mesh screen (Tyler type). The grinding at our large concentrator is at present carried closely to this point.

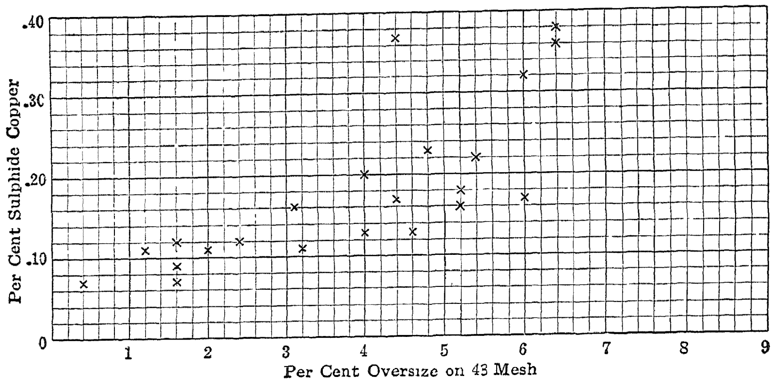

FIG. 18.—RELATION BETWEEN COARSENESS OF FLOTATION MACHINE FEED AND COPPER SULPHIDE ASSAY OF CALLOW TABLE TAILING.

OPERATION OF LARGE CONCENTRATOR

Flow Sheet of Callow Sections

The first four sections of the Commercial Concentrator of the Inspiration company were equipped with Callow flotation machines. The flow sheet followed in these sections is illustrated in Fig. 19. As will be seen from this illustration, the ore, after having been reduced to the desired fineness, and having been oiled in the Marcy ball mills, is sent to 8 (in other sections 16) Callow cells. The tailings from these primary cells are sent to a drag classifier designed by Mr. Burch, which effects a separation of sand and slime. The slime is retreated in 16 (in other sections 8) additional Callow cells, while the sand is sent to a hydraulic classifier and then to tables. The concentrates made both on the primary and the secondary Callow cells are retreated in a group of four Callow cleaning cells from which the tailings are returned to the head of the primary Callow cells. On the tables (Deister Machine Co.) a separation is made between concentrates and tailings, but no middlings are produced and no part of them is reground. This, however, is probably only a temporary arrangement, the idea being that a suitable regrinding and reconcentration process will be installed later, after having been worked out by extensive tests.

Flow Sheet of Inspiration Sections

Thirteen sections of the Inspiration concentrator are equipped with Inspiration flotation machines. The flow sheet of these sections is represented in Fig. 20. The ore, after having been crushed and oiled, passes to the head of the flotation machines and traverses their whole

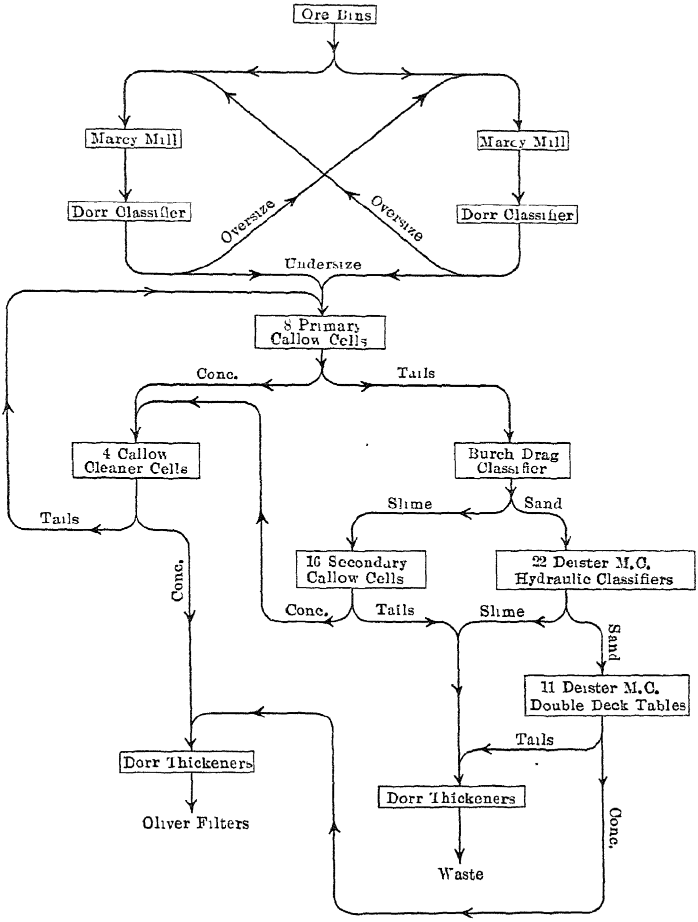

FIG. 19.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WITH CALLOW FLOTATION MACHINES.

length. The resulting tailings are split, by the same kind of a drag as mentioned before, into a sand and a slime product. The slime product is run to waste. This is thought permissible because, after having passed through the whole length of the flotation machine, the slime has been impoverished to such an extent that retreatment seems unnecessary

The sand product undergoes the same retreatment as that described for Callow sections.

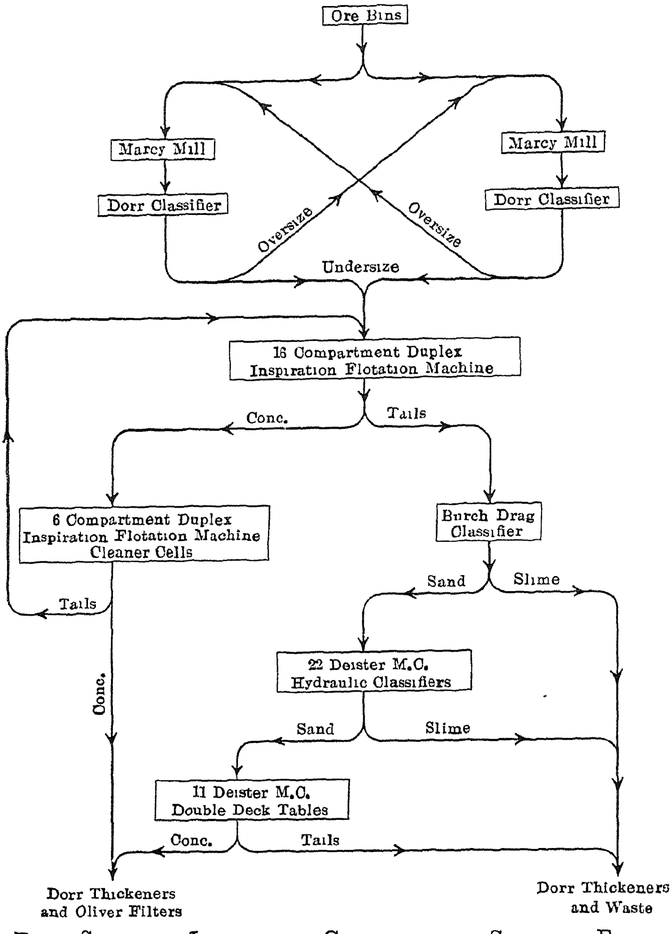

FIG 20.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTIONS EQUIPPED WITH INSPIRATION FLOTATION MACHINES.

Inspiration Flotation Machine of Steel Construction

Fig. 21 shows the drawing of an Inspiration flotation machine of steel construction as designed by Mr. Burch's office. At the end of each section of eight compartments, a pulp overflow is provided which regulates the level of the pulp in the preceding compartments of the flotation machine. The subdivision into two sections of eight is made in the interest of closer regulation of the pulp levels. It had been feared that if no such subdivisions were made, and the regulation of the levels accomplished at the tail end only, appreciable fluctuations might take place in the com-

partments near the feed end due to changes in the volume of pulp treated, and might prove highly undesirable.

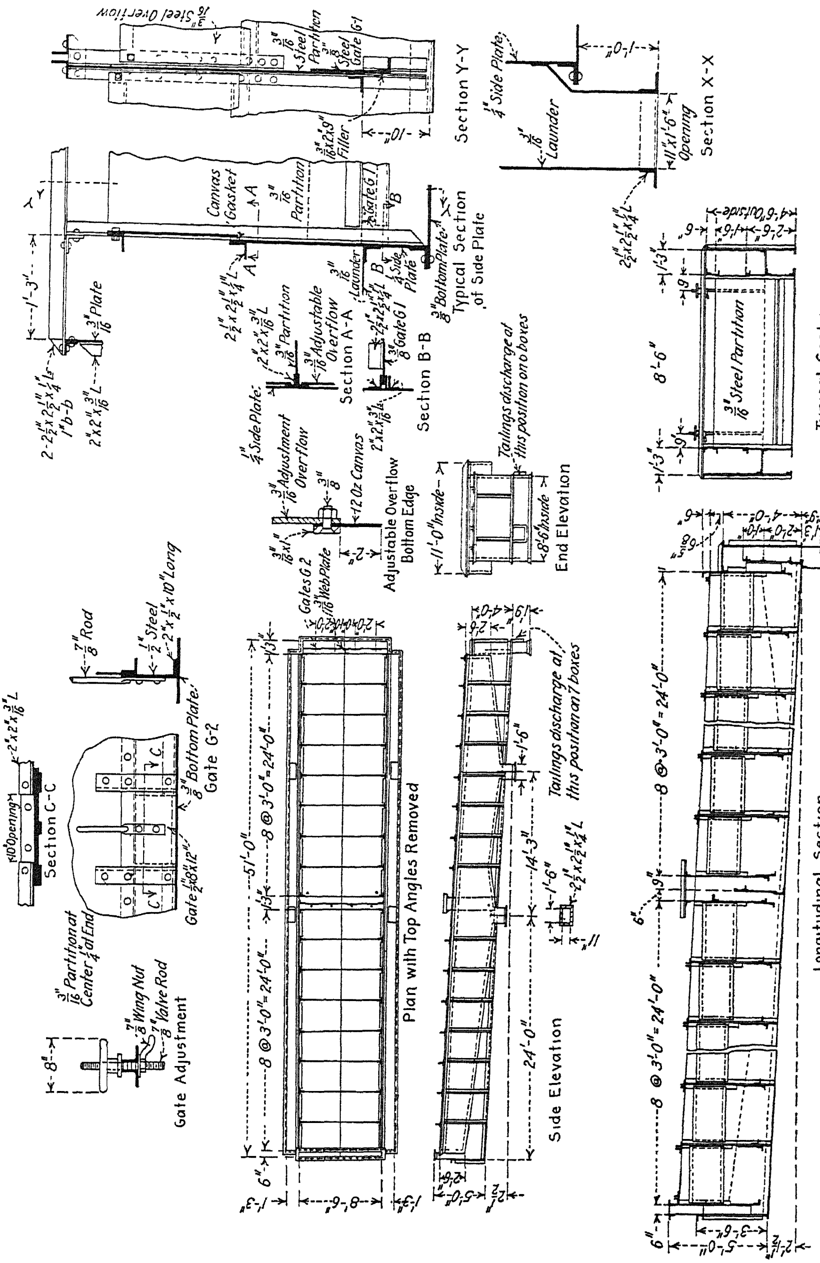

FIG. 21.—INSPIRATION FLOTATION ROUGHER CELLS OF STEEL CONSTRUCTION.

One point in the construction of the Inspiration flotation machines in which we take considerable pride, is the method in which the air is

introduced. The machines have solid bottoms; all pipe connections are made from above and are at all times visible. The porous bottom rests on

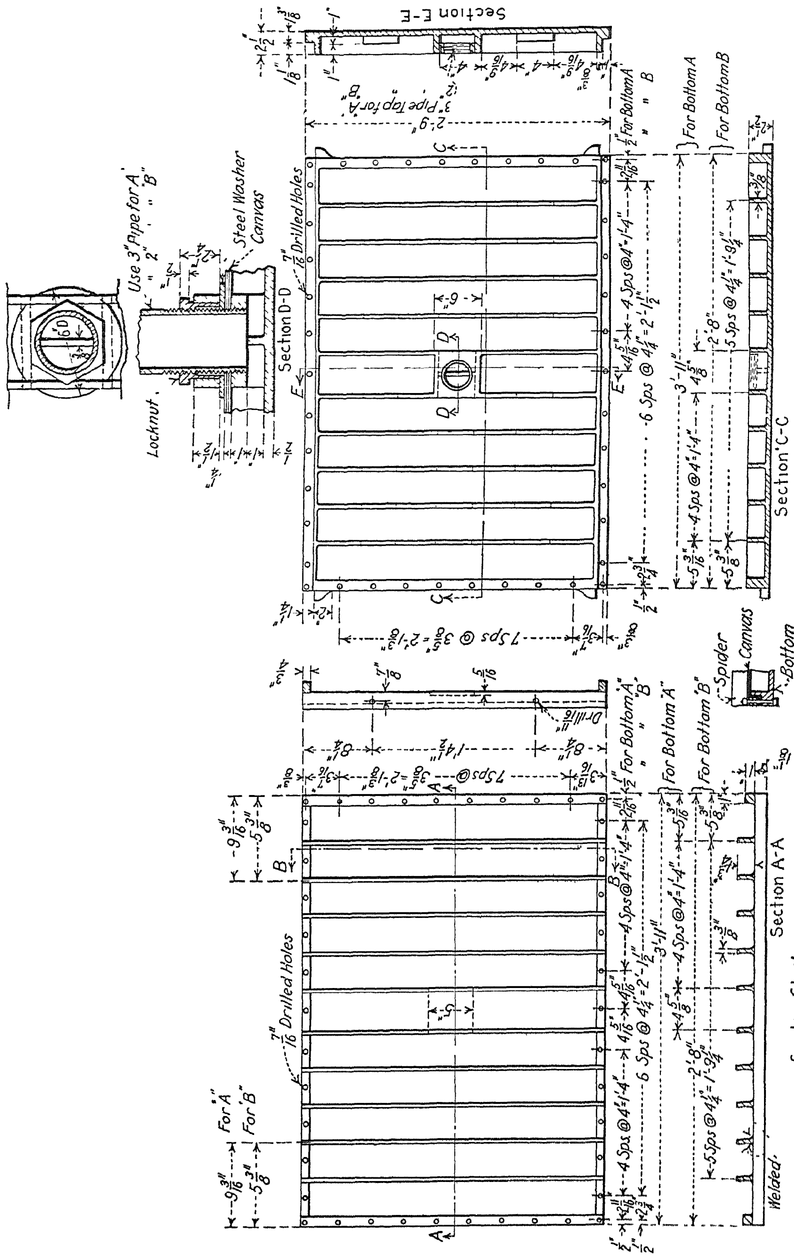

FIG. 22.—INSPIRATION FLOTATION MACHINE BOTTOMS.

top of the regular bottom of the machine and is introduced from above. It is illustrated in Fig. 22. As will be seen, it consists of a lower air

chamber having partitions running lengthwise, which, however, do not extend down to the bottom of the air chamber and, for this reason, do not interfere with the interchange of air between the narrow compartments thus formed. A grate, which forms the upper part of the porous bottom, can be fastened to the lower part by a number of bolts. Before putting them together, a porous medium (for instance, canvas) is placed between the two pieces. The air enters through a pipe from the top which screws into the lower casting. Arrangements are made, of course, to secure an air-tight joint where the pipe passes through the porous medium. When assembled, the bars of the upper grate form channels in the lengthwise direction of the machine. The bars are the only parts that protrude above the canvas and, on account of their arrangement longitudinally, they do not interfere with the passage of pulp through the machine.

The cleaner cells consist of six compartments in series. Both rougher and cleaner cells are divided by a partition in the middle, running the whole length of the machine. The operator is, therefore, enabled to throw all the feed on one side of the machine when desired, thus permitting repairs to be made on the compartments of the other side whenever necessary.

In the beginning, it had been supposed that throwing all the feed on one side of the flotation machine might seriously overload that side, and for this reason, in the first sections installed, provisions were made to permit removing and replacing the porous bottom from any one compartment without interrupting the flow of pulp. As far as lifting out the bottom is concerned, this, of course, can be done in any case, but while one of the bottoms is withdrawn sand will pile up to a certain extent in the middle of the compartment. To remove this sand before returning the porous bottom, a system of perforated water pipes was installed under the air chamber. They allow the washing out of the bottom of the machine by the application of water pressure.

Minerals Separation Section

One section of the concentrator is equipped with machines of the Minerals Separation Co., Hebbard type. The flow sheet, as illustrated in Fig. 23, is practically the same as the one shown in the Inspiration sections. The only difference is that two cleaner cells in parallel are provided in place of the one cleaner cell of the Inspiration type. The drawings of the machine are reproduced in Fig. 9.

Filter Plant

It is necessary to reduce the ore to a certain fineness before the mineral contents of the feed can be treated by flotation, and as fine concentrates

have the characteristic of retaining water with greater tenacity than coarse concentrates, a filter plant is a necessary adjunct of a flotation plant. In the Inspiration mill, both the flotation and table concentrates are sent to Dorr thickening tanks, five of them being 60 ft. in diameter and three of them 80 ft., representing a total area of about 29,217 sq. ft.,

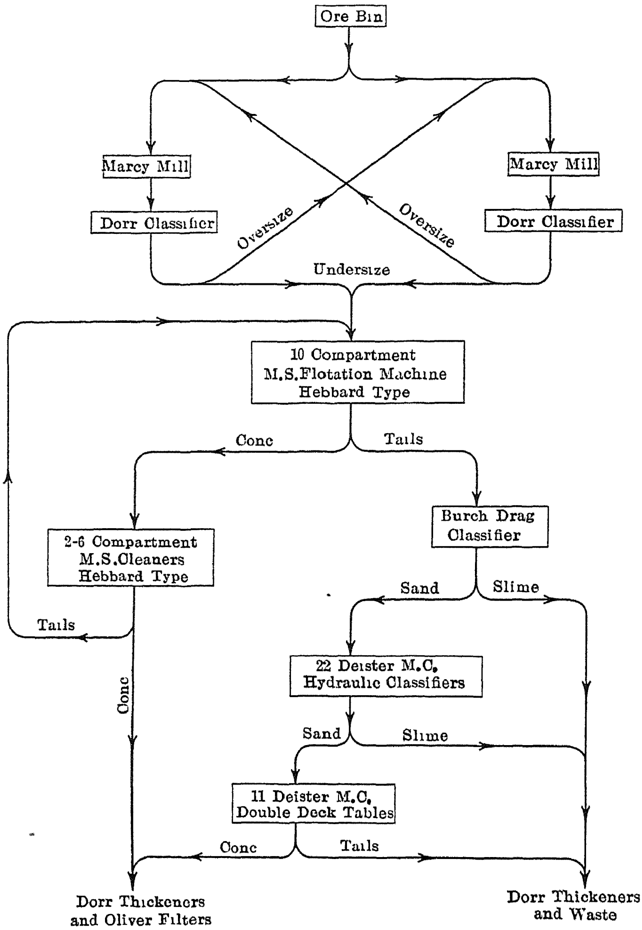

FIG. 23.—FLOW SHEET OF INSPIRATION CONCENTRATOR SECTION EQUIPPED WITH MINERALS SEPARATION FLOTATION MACHINES.

or, as the daily production of concentrates amounts to about 600 tons, a settling area of around 48.7 sq. ft. per ton of concentrate sent to the settling tanks. The settling tanks are each provided with a double ring of high baffle boards to prevent the foam that forms on the top from contaminating the overflow. The settling used to be carried out in two steps, some of the tanks serving as preliminary settling tanks, while

others resettled the overflow from the preliminary tanks. At present we operate all the concentrate settling tanks in parallel. The spigot discharges feed six Oliver filters. The ratio of solids to water is about 1.65 to 1 at this point. The filters reduce the concentrate moisture to an average of about 17 per cent. of the wet pulp. This figure varies, however, within rather wide limits from day to day. The moisture contents of the concentrate seem to depend in the first place on the percentage of insoluble material contained in the concentrates, as will be seen from the graphic representation of the relation between these functions in Fig. 24. It will be noted that a rise in the percentage of insoluble matter generally corresponds to a rise in the concentrate moisture and *vice versa*. A moisture of 17 per cent. is, of course, in excess of what seems desirable in view of the fact that the smelter penalizes water contained in the

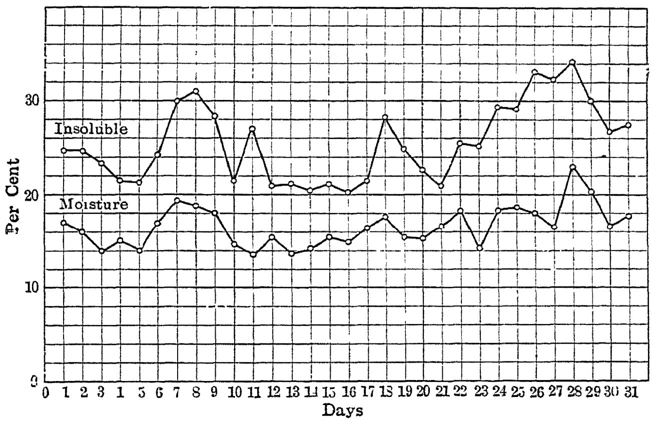

FIG. 24.—CURVES SHOWING RELATION BETWEEN MOISTURE CONTENTS AND INSOLUBLE MATTER IN CONCENTRATES. OCTOBER, 1915.

concentrates. For this reason, several attempts have been made to reduce the moisture. The first plan followed was to inject steam into the filter tank, with the object of heating the pulp. An appreciable increase in the capacity resulted from this, while no marked reduction of the moisture contents took place. The same effect was obtained by the addition of slacked lime, which we tried lately in our Oliver filters. It shows itself very readily in the formation of a thicker filter cake, thus causing a larger output of the filters. We have, however, not been able to effect a reduction in moisture by the use of lime. We have adopted the addition of lime to the filter pulp in regular operation on account of its decided benefit as far as the capacity is concerned. As a rule, five filters can handle the whole output of the concentrator; that is, each 12 by 12-ft. Oliver filter has a capacity of about 120 tons in 24 hr.

Results Obtained in Commercial Concentrator

On account of the fact that the concentrator operated by the Inspiration company has only just left the stage of construction, it is not possible to give results obtained in the operation of it as a whole. As, however, an interval of several months has elapsed between the starting of the first and the 18th section, we are in a position to give at least preliminary results derived in actual operation on a fairly large scale.

Tonnage

Each of the sections of the Inspiration concentrator has a rated capacity of 800 tons; at present most of the sections exceed 900 tons actual capacity. Ordinarily, this tonnage is treated on one Standard duplex Inspiration machine consisting of 16 double compartments, while the concentrates are cleaned on a cleaner cell with six double compartments. The size of the roughing compartments is 3 ft. by 4 ft. 3 in.; the size of the cleaner compartments is 3 ft. by 3 ft. Accordingly, the total area of combined rougher and cleaner cells is 516 sq. ft. Assuming the tonnage handled to be 800 tons, this makes a tonnage treated of 1.55 tons per day per square foot of surface. In the course of construction, the erection of flotation machines did not always keep pace with the erection of the grinding mills. For this and other reasons, it has frequently been necessary to combine the product from the two grinding sections in one flotation section; in other words, to throw an overload of 100 per cent. on the flotation machines. Our experience is, that overloading the machines by 100 per cent., or raising the tonnage to 3.10 tons per square foot of surface, does not increase the tailing losses very materially.

It follows, therefore, that the machines handle an overload without serious additional copper loss, and it has been our experience that there is no mechanical trouble whatever, connected with overloading the flotation machines to such an extent. We have also had to resort to overloading the Callow cells, and have likewise found that the recovery does not fall off seriously.

For the Minerals Separation machine of the Hebbard type, which is installed in our Section No. 10, we have no figures of a similar character.

Air and Power Consumption

We have found that three blowers, of a capacity of 8,000 cu. ft. per minute each, are sufficient to supply air for six sections of flotation machines. Accordingly, one section requires 4,000 cu. ft. of air per minute. As the flotation machines in an Inspiration Section have a porous surface of 340 sq. ft., it follows that the air consumption per square foot of porous

surface is 11.8 cu. ft. per minute; the maximum pressure required is 4½ lb. at the blowers and practically the same at the flotation machines, as there is no serious loss of pressure in the air-conduits. The power consumed for furnishing air has been determined carefully by wattmeter readings of the motors furnishing power for the blowers. The actual power consumed is 87.5 kw. per section; or, figuring again with the rated section capacity of 800 tons, 2.63 kw.-hr. per ton of ore treated. We have not been able to make separate tests for the Callow and Inspiration sections, but assume that the air consumption is approximately the same in both. We have no figures yet for the Hebbard type machine. To get the total amount of power consumed in connection with the flotation machines, a small amount of power, probably about 10 kw. or 0.30 kw.-hr. per ton of ore, has to be added to the power consumed for the production of air on account of the fact that the cleaner tailings are pumped back to the rougher cells and retreated, and that in the Callow sections the slime overflow of the drag classifiers is repumped to the secondary flotation cells. In the Inspiration sections, 0.425 sq. ft. of actual open porous surface and 0.645 sq. ft. of flotation machine area are provided for the treatment of 1 ton of ore in 24 hr.

Floor Space

The total floor space of the flotation floors amounts to 25,200 sq. ft., or 1.75 sq. ft. per ton of ore treated in 24 hr. (rated capacity). The total floor space occupied by the concentrator proper (not including, however, the crushing plant located at the mine) is 81,900 sq. ft. or 5.68 sq. ft. per ton of ore, and the total floor space, if the settling-tank installation outside the main mill building is also included, is 190,877 sq. ft., or 13.3 sq. ft. per ton of ore (rated capacity).

Water Consumption

The water added to the pulp where it reaches the flotation machines is about 3 tons of water for each ton of solid pulp. The subsequent treatment, in our case, requires a further addition of nearly 3 tons of water per ton of dry ore. However, as it is carried out on pulp that has been deprived of its slime contents by passing it over drag classifiers, the water thus entering the table tailings can be easily removed again. This is accomplished in our case by settling the table tailings separately from the slime tailings. The settling of the sand tailings is effected in a settling tank of home construction, consisting of a rectangular box with a sloping bottom into which a number of Caldecott cones are inserted. The resulting overflow is not quite clear, but is made so by resettling in three 60-ft. Dorr tanks.

Out of the total quantity of water actually added to the ore during its passage through the mill, amounting approximately to 6 tons of water per ton of ore treated, as mentioned above, nearly 3 tons, *i.e.*, the amount added during the table treatment, can, as explained above, be reclaimed rather easily. For the reclamation of the rest of the water, settling ponds, of the well-known construction in which the reservoir is formed by building a retaining wall of sand across a gulch, are resorted to in addition to Dorr thickening tanks.

Recovery

The recovery obtainable by the application of our milling process is determined entirely by the composition of the ore, *i.e.*, by the relation between sulphide and oxide copper contained in the ore. Our average sulphide copper extraction has been 90.39 per cent. for the months of March, April and May, 1916, the last months for which figures were available at the writing of this paper. A certain recovery of the oxide-copper minerals, especially carbonates, is made in the flotation process as well as in the gravity concentration process. The percentage of such mineral recovery is low, probably about 25 per cent. For this reason, the recovery obtainable on ore containing a high amount of oxide, such as surface ore, is correspondingly lower. We have worked in the laboratory with the object in view of increasing the oxide recovery; for instance, by adding certain chemicals to the flotation pulp. An account of the results obtained will be found below, but we have not yet applied this method to an operating scale, nor have we decided on using one of the other methods applicable for this purpose, such as leaching.

Table 5 gives average screen analyses of the feed and the general tailings of the Inspiration concentrator for the months of March, April and May. A segregation is made in the copper assay between sulphide and oxide copper, because, considering the present stage of the art, we feel satisfied with our mill work whenever the sulphide copper content of the mill tailings is low. As will be seen from the tabulations, a better recovery is made on the −200 material than on the coarser constituents of the ore, which proves the point that for ores of the character of Inspiration ore sliming is not to be feared since the introduction of the flotation process.

Oil Consumption

Experience has shown that in the flotation treatment of our ores, we consume flotation agents up to 1½ lb. per ton of ore. At present, the oil mixture contains about 95 per cent. crude coal tar and a little less than 5 per cent. of oils derived from the dry distillation of wood.

The different tars that we have tested during the operation of our mill have shown greatly varying qualities as far as their flotation value is

concerned. The first tar that we tested and made use of was home-made from domestic coal, and happened to be a very serviceable flotation agent. Since that time, we have tested tars from several States and have found some that are suitable for our purposes, while others are less so, and still others even entirely unsatisfactory. We have obtained satisfactory tar products from New Mexico, Colorado, Missouri and Illinois. These States furnish at present as much as we need for our consumption. For awhile, it seemed possible that we might have to import from a long distance the large quantities of tar that we require. During that period we tried to find substitutes, and looked especially toward the utilization of fuel oil for this purpose, but we have not been able to get as good results with any kind of fuel oil as with crude coal tar.

TABLE 5.—*Average Screen Analyses for the Months of March, April and May,* 1916, *of Flotation Feed and General Tails*

	Flotation Feed				General Tails							
Mesh	Per Cent. Weight		Copper Contents		Per Cent. Weight		Sulphide Copper Contents		Oxide Copper Contents		Total Copper Contents	
	Cum.	Indiv.	Per Cent.	Grams	Cum.	Indiv	Per Cent	Grams	Per Cent.	Grams	Per Cent.	Grams
+ 65	9 5	9.5	0.45	0.042	9.5	9 5	0.18	0.017	0.12	0.011	0.30	0.028
+100	21.2	11.7	0.86	0.101	21.2	11.7	0.19	0.023	0.14	0.016	0.33	0.039
+150	33.5	12.3	1.91	0.235	33.5	12 3	0.11	0 014	0.19	0.023	0.30	0.037
+200	39.2	5.7	2.69	0.154	39.2	5 7	0.14	0.008	0.19	0.011	0.34	0.019
−200	....	60.8	1.85	1.125	....	60.8	0.06	0.036	0.47	0.286	0.53	0.322
Totals	....	100.0	....	1.657	....	100 0	....	0.098	....	0.347	....	0.445
Assay direct	....	....	1.62	....	....	.. .	0.101	....	0.318	....	0.419	
Oxide	....	....	0.36									

Our experience is, that we can get along with coal tar alone. It is beneficial, however, to add wood-distillation products in small quantities, for instance, those containing pine oil. While coal tar makes a very strong and heavy froth, such as appears to be required to keep coarse mineral particles in suspension, the wood-distillation products have the characteristic of producing a multitude of froth bubbles, such as seem necessary to furnish the large surface required to save the very fine mineral particles. Because the finer ore particles (slime), on account of their small diameter, expose a large surface, it is evidently necessary to produce a correspondingly large surface of froth in order to save them by flotation.

Operating Cost for Flotation

The number of men necessary for the operation of large flotation machines is remarkably small. At the Inspiration plant, one operator supervises four sections of flotation machines. Two Mexican helpers assist him in washing the bottoms, thus insuring a free passage of air through the porous medium. At the prevailing high prices of American and Mexican labor, this means an expense of more than 1.0 c. per ton of ore treated. The total expenses representing flotation proper were as follows for the months of March, April and May, 1916:

	Cents per Ton
Labor	1 62
Flotation oils	1 65
Other supplies	0 35
Power	2.14
Total	5 76

The subsequent table treatment of flotation tailings, the filter treatment of the concentrates and other operations connected with the process of concentration, belong more or less to flotation treatment, and their expense should also be considered when the cost of the flotation process is to be established. The total milling cost, exclusive of crushing and grinding, has been for the past few months in the neighborhood of 20 c. When the cost of crushing and grinding is included, the cost is about 40 c. per ton of ore. Royalties for the use of the flotation process are not included in any of these cost figures.

Discussion of Results Obtained

The criticism has occasionally been raised against the Inspiration company that they were slow in deciding on the design of a concentrator. The reason for such slowness was, of course, that the development of the flotation process coincided with the period during which the fundamental points in the design of the concentrator had to be settled. The management, therefore, thought it best to be very conservative in installing standard concentrator equipment, which, in case the new process held what it seemed to promise, might prove to be entirely superfluous. The main point of interest, therefore, is whether it was wise for the company to wait instead of installing a gravity concentrating plant as had been originally considered.

As far as the comparison of a flotation plant, as at present operated by the company, with the gravity-concentration plant, as originally considered, is concerned, the recoveries actually obtained in the company's flotation plant are so much higher than those indicated by tests in the

original test mill as being obtainable in a gravity-concentration plant on ore of the same character, that there cannot be the least doubt which system is the better.

In discussing the second question which comes up in this connection, whether, for ores of the character of Inspiration ore, gravity concentration supplemented by flotation would be preferable to the simple flotation process that has been adopted by the Inspiration company, the crucial point is this: Can the slime be reduced to lower copper contents by a combined process than by the process followed here? The reduction of the copper in the sand is, as millmen know, simply a question of fine grinding combined with suitable concentration.

In support of the principle followed by the Inspiration company, I desire to call attention to the screen analyses of our general tailings represented in Table 5. They show that the sulphide copper left in the −200-mesh material is very low.

In addition, the floor space required for a plant like this is materially smaller than for a plant of the combination plan, which also means that the construction cost is much lower; and, further, the operating costs are low by reason of the greater simplicity of design. I hope these facts are sufficient to decide the point at issue as far as the ore of this district is concerned.

I hope that our neighbors of the Miami company, whose mill was built before the flotation process was known, will uphold me in my statements. If this point is admitted, it will endorse the principle followed in the design of the Inspiration mill, of doing away with reduction in stages and concentration in stages.

It must be granted that the settling of flotation-concentrate pulp requires extensive floor space, as shown by the figures referred to above, and that settling and subsequent filtering absorb a certain fraction of the mill operating costs. However, as a ratio of concentration in the Inspiration mill is 25 into 1, only $\frac{1}{25}$ of the ore is thus treated. The actual expenses increase, of course, proportionately for ore which does not permit concentration with such a high ratio as the ore existing in the Inspiration mine. Therefore, there will be a point beyond which a different system of concentration is preferable. Just what the critical ratio of concentration is, must be determined by calculation and testing based on individual conditions.

Prospects for Future Development of the Flotation Process

The flotation process is in its infancy. For this reason, therefore, our concentrator must be necessarily in the first stages of its development. In what direction future changes may take place, is perhaps indicated by tests which have been made partly on a laboratory scale and partly on a

somewhat larger scale, but which have not yet been incorporated into our regular milling process. Of these latent developments, I will try to give an outline in the following:

Porous-Bottom Experiments

The porous bottom is, as one may imagine, the most essential part of a pneumatic-flotation machine. Our experience with the porous bottoms of the different constructions brought out very clearly the principal difficulty attached to them, which is, that the pores have a tendency to diminish in size gradually and thereby to retard the passage of air through them. This tendency was more pronounced in the solid porous bottoms employed in the Flinn-Towne flotation machine than it was, for instance, in those of the Callow type, although the latter also show a tendency in this direction. Our first supposition was that the choking was due to the fact that the air entering below the blankets carried particles of dust, which would settle in the fine pores and reduce their area. Indeed, a canvas blanket will, after a certain length of service as a porous medium, always show some ring-shaped spots of dark color opposite the air inlets, clearly indicating that a deposition of dust particles on the blanket actually does take place. To make sure of this point we cut out round disks from a Callow blanket that had been used for some time and investigated their porosity by using them as porous bottoms in a glass tube standing in a vertical position. Air under pressure could be applied to an air chamber located underneath these disks, and the air passing through the porous blanket could be measured by a gas meter. The quantity of air discharged through the porous medium offers a measure of the porosity of the blanket, and for this reason, the velocity or speed with which the counter of the gas meter revolves, gives an indication of the porosity of the porous disks being tested. To our surprise, we found that the darkest points of the blanket were not those of lowest porosity. On the contrary, the points farthest away from the air inlet showed the greatest tendency to choke. An explanation of this paradoxical behavior seems to be offered by the fact that an air blanket is kept in a state of more or less agitation near the air inlet (in the Callow machine this happens to be a point remote from the places where it is held rigid) while farthest away from this point the blanket assumes a state of comparative rest. Incrustations, due perhaps to the presence of soluble salts in the water in conjunction with fine slime, always form to a greater or less extent in the top layer of the blanket. Evidently, the agitation counteracts the formation of the incrustation, while there is no such counteracting influence in the portions which are essentially at rest. For this reason, we concluded that a solid porous material is not suitable as a diaphragm in a flotation machine of the pneumatic type, if a bottom of long life is required. As a matter of

fact, the experience of everybody who experimented with solid bottoms seems to have pointed in the same direction. Mr. Cole for a while tested out carborundum tubes in his machine. We tried carborundum stones in the flotation machine of the Inspiration type and abandoned them, and I believe that even Messrs. Flinn and Towne have, in the meantime, given up the solid bottom of their original design.

A necessary condition for a serviceable flotation bottom appears, therefore, that the porous medium be of a flexible nature. The 4-ply canvas stitched every half inch or so which Mr. Callow's first cells contained and which we have used for considerable time in the Inspiration machines, seems to answer this purpose fairly well. We find, however, that to keep it in good working condition and prevent incrustations from forming on the top, we have to clean it frequently. This is done by dipping an iron pipe connected with a water hose into the compartments and sweeping the canvas bottom with the jet of water discharging from the lower end of the pipe. The canvas blankets seem to last for about 6 months at the most. As they are inexpensive, the replacing of a bottom after that time is not a serious item in the operating costs. The giving out of the canvas is due to the wear caused by the frequent cleaning. The top layer wears out first, the holes created by the stitching forming nuclei for the formation of larger holes. By the time the top layer has a number of holes the canvas blanket is generally discarded. In the interest of greater economy, we intend giving up interstitching the layers of canvas. We are trying to decide whether it is better to use single sheets of thicker fabric or to use canvas similar to the kind that we have been using and to put several layers on top of one another without interstitching them. The latter has the advantage of requiring the discarding of only one layer, when it becomes defective.

There will always be some tendency to form incrustations so long as canvas is used for flotation mediums. Their formation will be entirely prevented only by substituting an altogether different material. We have made experiments in this direction. One of my former assistants, R. M. Haskell, deserves credit for suggesting them. For instance, we substituted for the canvas blankets, thin rubber sheets perforated with a multitude of needle holes and obtained an excellent froth. The objection to their use is that their life is limited. When sheets of rubber of an increased thickness are used, the needle holes require too much pressure to form openings of sufficient size for the passage of air, and to make a thick rubber sheet suitable for this purpose, slits several millimeters long have to be substituted for needle holes. We have had one or two rubber bottoms of this design in operation, but just at present we are not ready to substitute them for canvas blankets. We also tried a blanket made from a material that goes under the name of sponge rubber and can be produced with rather fine texture. We have not been able,

however, to obtain lastingly good results from the use of this medium. Furthermore, we tried a woven fabric containing rubber threads in one direction and threads of cotton or the like in the other direction and a rubberized canvas made by the Goodrich Rubber Co. We are not prepared to use any of these materials on a commercial scale.

The advantage of rubber should be, in the first place, that on account of its smoothness it would have less tendency than canvas to permit the formation of incrustations. Besides, an elastic medium should have the additional advantage of avoiding the danger of catching small sand or slime particles in the pores of the medium, as an expansion of the medium (that may be effected, for instance, by increasing the pressure) would widen the pores and remove such particles. We think that our experimental work in this direction is encouraging.

Raising the Grade of Concentrates

The recovery that it is possible to effect in a flotation plant depends largely on the grade of concentrate desired. With a low grade of concentrate, low tailings can be made, but when a high grade of concentrate is stipulated, increased tailing losses cannot be avoided. A question that suggests itself in this connection, and which we have tried to answer by laboratory experiments is, "How can we raise the grade of our concentrates—that is, reduce the percentage of insoluble matter contained in them—without entailing additional copper losses?" We know from laboratory experiments that this can be done by expensive methods—for instance, by heating the solutions—but such a procedure would be undesirable from an economical standpoint. Experience has shown us that concentrate produced in the first compartments of the cleaner cells is always freer from insoluble matter than the concentrate produced in the last compartments. The problem then resolves itself into finding a suitable cleaning process for the concentrate from the last compartments of the cleaning cells. By treating this low-grade concentrate hot, with the addition of caustic soda, we have been able to separate it into a high-grade concentrate and fairly low tailings. This method necessitates only the expense of heating a small fraction of the pulp and may be a commercial possibility.

Recovering Carbonates by Flotation

Another subject on which we have spent considerable time in our laboratory is the problem of recovering copper carbonates by flotation. When we started our flotation plant, we discovered, to our astonishment that the machines not only saved a high percentage of copper sulphide but that they also recovered some of the carbonates. Ever since that time, we have tried to find means of improving the carbonate recovery.

In the first place, we studied all of the oils that seemed to have a tendency to cause the flotation of such minerals. Later on, we tried other means in addition to the variation of the oils. One way in which copper carbonates and similar minerals might be recovered was outlined by Alfred Schwartz in his United States Patent No. 807,501. The process consists in first artificially producing a sulphide coating on such oxidized minerals by the introduction into the pulp of soluble sulphides, and then adding suitable "oils" and effecting the flotation. If it were possible to thus chemically produce coatings of sulphide identical with the surface of the minerals formed by nature, this process would work well, as evidently the nature of the surface is the only characteristic that determines whether a mineral will float or not.

The Minerals Separation Co. owns a number of patents covering this subject. Their English Patent No. 26,019, issued to Sulman and Picard, describes the flotation of oxide copper minerals by similar means.

I am not aware that equivalent patents have been issued in the United States. The English patent in question is of a later date than the Schwartz patent above mentioned. The representatives of the Minerals Separation Co. have experimented more or less extensively with this system, while demonstrating their machine to the Inspiration company. As far as I know, they have not proven its practicability. In the course of their experiments, they tried the application of sodium sulphide and sodium polysulphide for this purpose. The latter was produced by treating sulphur with hot caustic soda. At the time these experiments were made, I was not familiar with the chemical action taking place, which, as much as I know now, actually results in the formation of polysulphide mixed with thiosulphates and other oxygen-sulphur compounds. The failure of their experiments, I therefore ascribed to the fact that perhaps a polysulphide, which they were anxious to make, was not actually produced. I proceeded to make sodium polysulphide by a method which I knew, that is, by the treatment of a sodium sulphide solution with sulphur powder. When we applied this reagent to some of our carbonate ores in laboratory flotation experiments, we noted that a good recovery was obtained. The composition of the compound was varied materially in order to find just what composition gives the best results in the flotation of carbonates. Our experience seems to indicate that sodium sulphide alone encourages the flotation of carbonates, but that sodium polysulphide, or sodium sulphide which contains more sulphur than would correspond to the chemical formula Na_2S gives better results. The addition of caustic soda besides the sodium polysulphide was found beneficial.

The question then arose as to why we succeeded in effecting the flotation of oxidized copper when the experiments of the members of the Minerals Separation staff failed. Tests along these lines brought

out the fact that the Minerals Separation compound when applied to our carbonate ores also worked successfully, but that it did not on our regular milling ore. Our own compound when added to our mill ore mixture increased the recovery of the carbonates, but evidently interfered with the sulphide extraction, and for this reason seemed to be of as little use as the compound of the Minerals Separation Co. When applying reagents of this character to tailings resulting from ordinary flotation treatment, with the point in view of effecting a sufficient sulphide extraction by the regular flotation process, and using the compound in question only for the purpose of increasing the carbonate extraction, we have found so far that the increase in copper-carbonate recovery over the one obtained without the addition of such chemical compounds is not worth going after.

But this is only a consequence of the fact that carbonates exist in very small amounts only in our milling ore and are partly saved by the ordinary flotation process.

There is no real difficulty about saving carbonates by the method mentioned, if they exist in quantities that make it worth while to save them. That copper carbonates can be recovered may easily be demonstrated by treating a deslimed feed in a series flotation machine. If at the point of the machine, where the sulphide recovery is nearly finished, sodium sulphide is added, the decidedly green color of the concentrates in the following compartments leaves no doubt on this point. The desliming of the feed seems to assist in the carbonate recovery.

It is of considerable (even if only theoretical) value to establish why sodium sulphide and polysulphide tend to increase the recovery of copper carbonates. A coating that might be expected to form cannot be detected. The concentrate resulting from the treatment of pure carbonate ore is decidedly green; besides, when an alkaline condition of the pulp is used there is very little, if any, tendency for any sulphide coating to form, and the alkaline state of the pulp is (as explained above) exactly the condition under which the best carbonate extraction results. Another point that seems to contradict the explanation of these results by the assumption of a sulphide coating is, that when we proceeded exactly as suggested by Mr. Schwartz—*i.e.*, when the application of soluble sulphide was followed by the addition of flotation agents and by the actual flotation—we seemed to obtain poorer results than when the procedure was reversed by applying the oil first and following with the application of some soluble sulphide, although the latter method would certainly seem less favorable to the formation of a sulphide coating, and perhaps for this reason has not been suggested by Mr. Schwartz.

Another theory that has been mentioned as an explanation of this phenomenon is that colloidal sulphur is formed by the solution of sodium polysulphide in water, which, as is known, is a good flotation agent. For

instance, it is pointed out in the United States Patent No. 1,140,865 taken out by Dr. R. F. Bacon of the Mellon Institute in Pittsburgh, that by setting free colloidal sulphur, say by the reaction of a soluble sulphide with sulphur dioxide, good flotation results may be obtained as far as the flotation of sulphides is concerned. To make the process available for the flotation of carbonates and other oxidized copper minerals, he suggests that a sulphide coating be first formed on the minerals, *i.e.*, to follow Mr. Schwartz's idea. Whether the colloidal sulphur by itself has a beneficial influence on the recovery of the carbonate (as has been suggested in explanation of our observations) seems rather doubtful when it is considered that we have obtained good results in alkaline solutions in which colloidal sulphur does not seem to separate out from polysulphide containing only a limited amount of sulphur such as was used in our tests. The full theoretical explanation of these facts must therefore be left to future investigations.

Recovery of Silicates

In our experiments with the object of saving the oxidized copper minerals, we soon found that we could save some of these minerals, while others were entirely refractory to the method above mentioned. To establish which minerals could be saved and which not, we attempted an analytical separation into carbonates and silicates. The chemical methods which we tried for the purpose of distinguishing between the two proved unreliable, however, and we had to resort to the separation by specific gravity (panning). The carbonates of copper (malachite and azurite) are heavier than gangue, and the silicates (chrysocolla) are lighter. The separation is rather difficult, owing to the small difference in specific gravity, and the results are therefore far from being altogether reliable, but they seem accurate enough to indicate that the method of saving carbonate copper above referred to is of value only for the recovery of carbonates and does not apply to silicates. This fact seems to be another corroboration of the assumption made above, that carbonates of copper do not float simply because of the formation of a thin surface coating of copper sulphide. It can easily be verified in the laboratory that silicates can be coated with copper sulphide fully as easily as copper carbonates. For this reason, if the filming theory is right, it should be possible to float silicates just as well as carbonates. There is no doubt that they can be floated by transformation into sulphides, only this transformation must not be confined to the surface, but must go deeper. Our experience is, that to effect a good recovery, it is necessary to acidify the pulp so strongly that practically all of the silicates of copper are dissolved and by the action of hydrogen sulphide or other soluble sulphides are transformed into the state of chemically precipitated copper sulphide.

In this form there is no difficulty about the recovery of the copper by flotation, but this procedure is not entirely without objection.

In case hydrogen sulphide gas is used, the acid combined with copper is regenerated. This tends toward a low acid consumption and a good copper extraction, on account of the fact that the treatment winds up with a small percentage of copper in solution and free acid present, both of which are desirable in the light of the law of chemical mass action. But hydrogen sulphide is not a desirable reagent. The fact that it is a gas and not a liquid introduces complications in the apparatus which are accentuated by the fact that it is poisonous and obnoxious otherwise.

Other soluble sulphides used in place of hydrogen sulphide will neutralize some sulphuric acid with the result that the acid consumption will be higher and the copper extraction lower than in case of hydrogen sulphide gas.

As far as acid consumption is concerned, it is pointed out that the free acid lost with the pulp may be settled out in ponds and re-used. However, the re-use of acid diluted to such an extent is a more serious problem than is generally realized.

The treatment of concentrates that are "colloidal," to a much greater extent than ores which mill men have been in the habit of calling colloidal, offers additional problems, which, however, may prove not to be as serious as they look.

Everything considered, I cannot see that the flotation treatment of oxidized copper ores after previously leaching them offers better prospects than straight leaching by decantation and precipitation by other methods.

General Theory

Very much has been published recently about the theory of the flotation process, and very many suggestions have been made that will probably prove valuable after it has been shown by critical tests how far they explain the facts.

It seems to me that an explanation of the qualities of the flotation oils is not as difficult as it might appear. The problem only seems so complicated because the flotation qualities of an oil or an oil mixture have not been separated into their components. In fact, it requires a combination of qualities to make a successful flotation oil. In the first place, the flotation oil has to coat the mineral particles. That there is a tendency for the formation of such a coating can easily be seen from simple experiments. For instance, if samples of copper sulphide (chalcocite), copper carbonate (malachite) and gangue (silica) of the same screen size are spread out on watch glasses and then moistened with a drop of coal-tar creosote, it will be seen that the drop of creosote soon disappears through absorption by the copper sulphide, while it takes a much longer time for it to be absorbed by the copper carbonate and a still longer time

with the gangue. On the other hand, when a drop of water is placed on the same minerals, it will disappear on the gangue first, later on the carbonate, and finally on the copper sulphide. This evidently proves that in a mixture of water and oil, the oil will attach itself with preference to the sulphide particles while the water will have the greater tendency to wet the gangue.

The second quality which at least is sometimes required of a flotation oil, is that it has to form a stable froth. In such a case, the stability may be secured by more firmly cementing together the mineral, air and oil. To accomplish this, oils are used which have a tendency to float finely divided gangue particles. The action is characteristic of the heavier pine distillates like pine tar and the lighter ones like turpentine if they are crude, unrefined products; in other words, when they contain some of the heavier distillates. I am not quite sure, however, whether the beneficial influence of oils of this group is not perhaps rather due to the fact that they remove colloidal material from the pulp and thereby increase the tendency of the minerals to float.

A third quality demanded of a successful oil mixture is that it must be able to produce a sufficient volume of froth. This property is exemplified best by oils of the soluble type—cresol, pine oil, alcohols and other substances. It can easily be proven that when oils of this type are used the water acquires the frothing qualities of the oils, although they may be considered insoluble. It may be demonstrated by shaking an oil of this character with water and permitting the oil to separate out again. It will be found that the water has acquired frothing qualities by undergoing this treatment. It is perhaps even likely that the soluble portion of the oils belonging to this group is the only portion that is active in this manner. The difference between the oils of group 1 and group 3 may be studied, for instance in a flotation machine of our type. It will be noted that the heavier mineral runs over the concentrate discharge largely in the first compartments forming a heavy, dark froth and the heavy insoluble portions of the flotation oil mixtures apparently go with it. Toward the tailings end of the flotation machines, most of this dark material has disappeared and the froth is lighter and of a more watery nature. The pulp, however, has not lost the quality of forming froth even after it gets to the last compartment of the flotation machines. This permits the conclusion that the frothing characteristics follow the tailings pulp. The water settled out in tanks and tailing ponds has decided frothing qualities. Such water behaves in a similar way to certain alcoholic solutions with which we are used to associate this characteristic, for instance, beer or champagne. The experience of mills using the flotation process, that when the tailings water is reclaimed the quantity of frothing oil may be considerably reduced, further supports the assumption that the formation of froth is caused by water-soluble substances.

Just how the surface tension of water and air must be modified to permit the formation of froth has been made the subject of some speculations recently published by different authors. We have also devoted some thought and a few experiments in our laboratory to this question. A discussion of these matters belongs to the realm of physics, however, and is outside the scope of this paper. But I might add this remark to the discussion of the subject of flotation oils, that most flotation oils not only have the characteristics of one group, but may at the same time possess those of another one. For instance, coal tar has qualities 1 and 3. For the flotation of our ordinary milling ore we do not require much of the quality of oil classified in group No. 2, and, therefore, can get along with coal tar alone. But we find it advisable to add to the coal tar more of the qualities characteristic of the third group, and for this purpose, we add about 5 per cent. of the total in the form of crude pine oil. In many cases it is found that the flotation oil has the characteristics of group No. 2 to such an extent that it is impossible to make clean concentrates. Various chemical means, such as the addition of acid or alkali, are used to counteract this.

Conclusion

In summing up I want to say that the fact that the Inspiration company has been able to design a commercially successful flotation plant and has found ways that hold out prospects of raising the plant to a very high state of efficiency, must be attributed to the policy followed by the company of spending great sums of money for the purpose of investigating the flotation process on a commercial scale. In carrying out these investigations, a close coöperation between laboratory and operating force helped us, I believe, more than anything else. I would like to give credit to each person who had a share in contributing toward the success of the work, but cannot do it, because the list would be too long.

Discussion

David Cole, El Paso, Texas (communication to the Secretary*).—I have read with great interest Dr. Gahl's painstaking paper giving us the details of development of flotation at Inspiration, and it seems to me that he has covered the ground so completely that there is little to discuss or criticize in connection with his subject. While it was in operation, the test mill in which the campaign was carried on was the Mecca of mill men and metallurgists. There was much to interest students of milling methods in addition to the flotation experiments, and readers of Dr. Gahl's paper will miss the information they hoped to get when the

* Received July 8, 1916.

"pilot" mill results were finally compiled, and I hope that the following remarks outlining from a distance some of the things that happened there will bring forth the rest of the story of the performances recorded in these important experiments.

Prior to the use of flotation methods in concentration it had long been recognized that the "unavoidable" losses of sulphides were in the slime which is inevitably produced in the grinding operations required to free the minerals to be separated. Classifying the feed prior to its final stage of treatment had long been in style in milling. This assisted the sand-handling machines and resulted in lowering the tailings made by them, but at the expense of the slime-handling department; and while little was expected of the latter, the wisdom of complicating the process by hydraulic classification was being seriously questioned. Indeed, at the time that the gravity method flow sheets were being developed for Inspiration ores, there was a minor revolution in milling methods impending involving the elimination of the hydraulic classifier. This change gave no promise of higher extractions but did promise greater simplicity and much less floor space per unit of capacity. Such mills promised to be less costly to build, less expensive to operate, and equally efficient. Wholesale concentration in relatively small space would get as good results as piecemeal concentration had been getting in the multiple operations of the spread out plants into which the porphyry treatment mills had degenerated.

The one great drawback was the slime. The desiderata of the engineer and the manufacturer were, on the one hand, to provide grinders which would produce the minimum of slime, and, on the other hand, improved "slimers" which would give the maximum recovery from the finest products. The technical press of the period reflected this state of things in the advertisements of those having the latest novelties to sell. There was much revamping of old ideas with some refinement, but nothing new. Tube-mill grinders were "taboo" for concentration, because they had a bad reputation as slimers. Automatic canvas plants were being exploited and very ingenious multiple-deck table devices were being offered as the remedy—the only possible remedy.

Looking backward no further than 1912, when Inspiration began to study its milling problems, we now see that we were without effective resource in treating real sulphide slimes. Mr. Callow's investigations and experiments had apparently demonstrated that some departure from the usual gravity practice would be advantageous and that a high recovery for that method would be possible on the granular material, but he threw small light on slime treatment. This was the situation when the Inspiration company was endeavoring to determine its mill treatment scheme.

After reviewing from every angle the results of experiments on the ore

and other information available, a modified flow sheet was finally crystallized by Mr. Burch and adopted by the management. Mill plans were drawn for what was to be a most highly developed 7,000 tons per day gravity process plant, and work was immediately inaugurated to carry out these plans. The mill site was selected, and much active work had been prosecuted before flotation (by this time being hastily tried in the old experimental plant) had so far won its battle that results could be viewed otherwise than probably too good to be true.

The apparent promises of flotation were extremely attractive, because the process would be simple and would solve the all-important slime problem. The process would have a greatly reduced number of stages; the plant would be much smaller per unit of capacity; the cost of construction per unit of capacity would probably be very much less; the use of water would be minimized; grinding would have to be carried further than usual, and would be the main item of milling cost, but this was not a very great handicap because sliming did not matter. What system of grinding would be best to use and what is the best machine? Would it be possible to parallel the small test-mill metallurgical results on a full-tonnage basis, and finally would it not be too hazardous to accept so revolutionary a process with so much experiment in its makeup?

The mine preparation program which had been settled upon would produce about 600 tons daily of freshly broken ore directly from the headings. This happened to be the rated capacity of a full-sized Minerals Separation unit which the flotation people were urging as a means of improving their extraction. If this ore from the headings were put in stock pile in the usual way, pending the completion of the milling plant, the ore would oxidize to some extent and besides would involve reclaiming expense later on. Why not mill it all as fast as produced, in a real "pilot" mill, wherein not only flotation problems, but grinding problems, power consumption, use of water, preparation of sticky concentrates, and other vital questions which might come up could be definitely threshed out under what would be regular commercial conditions on full-sized machines? In this size of "pilot" operations, labor would be used economically and production would almost, if not quite, pay all of the current expenses, except the mining cost.

This program was adopted. That the decision to carry it out was a wise one is shown in Dr. Gahl's paper, and that it paid its way is shown by Mr. Mills' annual report for 1915, in which he says: "Contrary to the usual experience, this test mill paid the cost of its construction, its operation expense, the present average mining cost on ores treated, and something besides, and has been written off the books."

Anybody with a real idea applicable to the problem could get a hearing and a tryout in this extremely practical commercial sized laboratory where sampling was in the hands of engineers and the results were com-

piled in a way to make them comparative and valuable. Thousands of dollars were spent by the company, by inventors, and manufacturers in demonstrations. Expense was subordinated. Heavy shipments by express were made when necessary to hasten the work, and much more than flotation was developed in this "pilot" plant.

Four different types of Symons crushers and pulverizers were tried. Three of these were marked and very interesting departures from ordinary practice. The company had purchased the Hardinge mill patent rights for Arizona and four forms of this mill were installed to determine the best to use, and these were kept busy during nearly the whole campaign. A long parallel tube mill was installed and run in competition with the Hardinge mill A high-speed Huntington-type grinder at one time attracted much interest. Hammer pulverizers of two different makes were also tested. Various linings and grades of flints were tried in the pebble mills. Steel balls in place of pebbles were advocated and a carload purchased.

In the latter part of the testing period the Marcy type of ball mill, especially designed for using iron balls larger than usual, and adapted to crush from breaker sizes to 48-mesh in closed circuit in one operation, was installed and perfected. This grinder proved capable of a greater range of reduction than had previously been thought possible, from an initial feed as coarse as 3-in. cubes. It is a ball mill pure and simple, having large capacity in small space. It makes use of a perforated diaphragm to keep the balls and charge inside of the mill until the latter will pass a 3/16-in. opening, and it has the equivalent of a peripheral discharge. An overflow classifier determines the finished size and the oversize is continually returned to the grinding chamber. This mill uses little water in the grinding chamber, so that its charge of ore is mortar like in consistency. It was quite successful. There was nothing in the Hardinge equipment to parallel it because the Hardinge mills were built for pebble mills and did not have feed scoops or openings adapted to handle as coarse a feed, and the linings would not stand up under the ball load. Would the Hardinge machines when built as a ball mill with the required strength, type of lining, size of feed and discharge openings, do as well? To wait for a mill to be made over or a new one manufactured would take too long, so the Marcy type was adopted and, contrary to what I think is the popular impression, the conical-type steel-ball mill *vs*. the Marcy-type steel-ball mill did not receive a tryout.

Recording electric instruments were installed so that accurate power records could be continually made while the various machines were being operated.

Several varieties of drag and rake classifiers and two types of vacuum filters were installed and records made. The efficacy of high-reduction herringbone gears driving ball and tube mills became a matter of interest

on account of the troubles that developed in them, and the reasons for these troubles, which would make a story by itself when discovered.

This testing work grew into a process of elimination and the scrap pile grew steadily. Some blasted hopes may be buried there, but it does not follow that all of the machines or materials that were returned to the sponsor or that found their way to the scrap heap were entirely unfit. It was necessary to choose and to discard, and that there is no acrimony in connection with the matter speaks well for the type of justice and judgment that prevailed. Doubtless the use of some discarded things might have answered as well, but it would be hazardous indeed to say that anything vital on the score of cost or recovery of values failed to receive recognition in the final selection. One of my mental offspring was among the fallen. It held out for a long time and I greatly appreciate the favorable mention which Dr. Gahl has made of it. I would have been pleased to pursue its development further, but I can not at this time see where its adoption would have saved any more copper or any more money in getting the copper than the ones that were selected, and I think this will apply to all the "late lamented" concerned in the campaign that was carried on. In other places and on other ores the conditions would not be the same. The ratio of concentration, or the crushability factors particularly, may so change conditions as to indicate the use of ideas that were discarded at the Inspiration test mill.

When the comprehensive plan of testing on a large scale was decided upon, I think Dr. Ricketts and Mr. Mills recognized that the records made might be extremely valuable to engineers and metallurgists, and I believe the matter of writing the story of the campaign was considered and that records were started with that ultimate end in view. Experience added to theory is more valuable than theory alone; the scrap heap more eloquent than the machine promotor, and the wornout unit in the "bone yard" is often more interesting than the one shown on the original blue print. It is seldom indeed that experiments are possible under such auspices and on such an extensive scale as those carried out in the Inspiration "pilot" mill, and I hope that Mr. Burch or Dr. Gahl, or both of them, will, with the approval of the Inspiration company, find time and inclination at an early date to prepare a paper or papers which will extend, through the *Transactions*, the value of these experiences to our members.

Referring to the final flow sheet adopted, I note that hydraulic classification had no place in the 600-ton "pilot" mill experiments which Dr. Gahl has so interestingly described, and I note that he has not referred to the reason for retaining this remnant of the old system of concentration in the flow sheet.

I note that 3 tons of water per ton of ore handled is required in the flotation operation and that 3 tons more is added in the subsequent table treatment which, of course, includes the hydraulic classification operation

practiced, and I presume that something more than one-half of the last 3 tons is added in the classifiers themselves; and since the water is clarified and returned, the addition of unnecessary water would entail expense for clarification and pumping which would not be required if less water were used.

In my work in the concentration of ores I have not been able to become very enthusiastic over hydraulic classifiers, and since flotation has come to the rescue of the slimed sulphide I find myself less enthusiastic than ever over their use.

When I was a lad in the Black Hills I was fascinated in watching a certain mountain spring in which polished micaceous particles glistening in the sunlight would be caught in the current rising from an orifice in the bottom of the sand funnel and be flirted to the surface, sail across the crater and fall upon the conical sides to be again methodically returned over the same route. It was interesting to watch the disturbance caused by dropping a handful of foreign sand and silt into the funnel and have it "classified" and washed clean, a new form of crater finally being established with the changes of average sizes retained. My first contact with hydraulic sizing in concentration was studied from that foundation. The spring took its time to do a good job; it worked the charge over repeatedly. All the silt went out quickly and the fines gradually went overboard with a rapidly decreasing ratio, until the crater would settle down again to its regular work of turning the mobile contents over and over in a new condition of equilibrium. But I soon learned that the beauty was all taken out of the process in its commercial application.

The process witnessed in the action of the spring was balanced, precise, and definite, and quite at variance with what we witness in watching through glass the operations going on in a "teeter chamber" of the metallurgical hydraulic classifier, which seems to me to be of little value except in its office of washing out the slimes which used to be the main source of the loss in treating unclassified material upon concentrating tables.

Following the thorough combing out of the slimed values as effected by the splendid flotation treatment that the pulp has previously received in the Inspiration final flow sheet, I am inclined to question the value of the subsequent classification by hydraulic means. I note that the flotation tailing is split into slime and sand at the drag belts, that there is very little slime left in the sand portion, and that what little there is (on account of the previous frothing) is completely devoid of value which the tables can save, as indicated by the fact that the main slime overflow of the drag-belt separators is discharged to waste without further treatment. It seems to me that it ought to be possible to eliminate the classifiers, and I would like to ask Dr. Gahl if there has been any trial to determine what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation.

RUDOLF GAHL, Miami, Ariz.—Since I wrote the paper on flotation which is in your hands, important developments have taken place, and, for this reason, I will try in a few words to bring it nearer up to date as far as the Inspiration plant is concerned.

You may have noticed that in the Inspiration concentrator, flotation is not solely relied upon for the recovery of the coarser material, but that flotation tailings are split into a sand and a slime product on drag classifiers. The sand product is treated on tables which thus supplement the work of the flotation machines.

Extensive tests have shown us that, if we wanted to, we could leave out these tables and substitute additional flotation machines, as they will make fully as good a recovery, if not better, than tables on the deslimed feed, but the treatment would be more expensive, especially in view of the fact that oils are required which cost more than those which we are now using in our main flotation plant. We have, however, decided to apply flotation treatment to our table middlings.

I would also like to add a few words regarding the treatment of oxidized copper ores. Although our experience shows that the addition of hydrogen sulphide and other soluble sulphides effects a very good recovery of copper carbonates with certain ores, we have not been able to prove that we could use it advantageously for the ore mixture which we are treating in our concentrator, and for this reason, have looked toward leaching for extracting the carbonate and silicate copper that we are losing now. Experiments in this direction are going on and are giving very encouraging results. It may interest you to hear that we intend to use limestone for the precipitation of the copper which goes into solution, as electrolytic precipitation seems to be out of the question on account of the diluteness of the resulting solutions, and precipitation by iron was rightly objected to on account of the unavoidable contamination of the water supply. We feel very hopeful about the success of our limestone precipitation which, if it holds what it seems to promise, will develop into a novel feature of copper metallurgy.

FREDERICK LAIST, Anaconda, Mont. (written discussion).—Dr. Gahl is assuredly to be complimented on the preparation of so well-written and complete a paper as well as upon the excellent work done under his supervision. One cannot but be impressed by the care and thoroughness with which all possible combinations were investigated, and the paper leaves one with the impression that the equipment finally chosen is without a doubt the best obtainable for the conditions prevailing.

I was specially interested in Dr. Gahl's description of the development of the Inspiration flotation machine, and its final perfection is certainly a credit to his keenness and powers of observation. The machine is

simplicity itself and strikes me as the logical development of the pneumatic type.

The relative merits of the impeller and pneumatic types of flotation machines have been the subject of much discussion and the selection of type is doubtless dependent largely on local conditions and on the characteristics of the ore undergoing treatment. We have always been of the opinion at Anaconda that whenever a neutral or alkaline treatment was used and the oils could be added to the pulp going to the ball mills, the pneumatic type had an advantage both as regards power consumption and installation cost. When, however, the treatment requires the use of acid as is the case on some copper ores and most zinc ores, the pneumatic machine loses much of its advantage. Obviously, the acid cannot be introduced into the ball mill and it is generally necessary to add it ahead of, or at the same time as, the oil.

Thus it becomes necessary to insert an agitator between the ball mill and the flotation machine as the pneumatic treatment alone is not sufficiently vigorous. The early pneumatic machine installations contained Pachuca tanks for this purpose. These, however, did not prove effective, for the reason that an emulsification of the oil is required, and not merely agitation. It therefore becomes necessary to use an impeller or some form of mechanical emulsifier, and the power required to operate this machine must be added to the power consumed by the flotation machines proper.

In Montana we find that our power consumption for emulsifying and flotation is about 0.24 hp. per ton as compared with 0.15 hp. for flotation alone at Inspiration. The capacity of an impeller-type machine is materially greater when the emulsification of the oil in the pulp is done in the ball mill. In this connection an interesting suggestion was recently made by Dr. Cottrell. He suggests making an emulsion of oil and water in a special emulsifier, such as made by the De Laval people, consisting of two disks running almost in contact. The oil and water is fed in at the center and is thrown out at the circumference. Thus the work of spreading the oil through the pulp might be done more efficiently than is now the case.

For some time it seemed to us that the main point to be considered in choosing between the two types of machines was power, and that this depended largely upon whether acid or neutral or alkaline treatment were decided on. Of late, however, it has seemed to us that the impeller type of machine has another advantage, which, I recall quite distinctly, was cited as a disadvantage when the first Callow machines were brought out. I refer to the toughness of the froth. Most of you doubtless recollect that one of the advantages of the pneumatic machines was supposed to be that the froth breaks down quite readily, thus rendering the mineral content of the froth easier of collection.

It is becoming more evident to us, however, that this apparent advan-

tage is actually the reverse, for the reason that the coarser mineral grains tend to fall back before they can be skimmed off and are thus lost or must be recovered by tabling. We are beginning to believe that the tougher froth is a distinct advantage of the impeller machine, and our belief has been considerably strengthened of late by tests made on a disseminated copper ore from South America. It was impossible to make as lean a tailing on the pneumatic machine as on the impeller machines, owing to falling back of the coarser mineral grains.

Francis S. Schimerka, Clifton, Ariz.—Regarding the proposed scheme of Dr. Gahl, to precipitate the copper from a sulphate solution by means of ground limestone, I had a discussion with him a few days ago. I called his attention to the fact that the result of this operation would be a low-grade, very slimy and voluminous precipitate consisting of the rather insoluble gypsum and basic copper carbonate both in highly hydrated form, which is difficult to handle and can be worked up only by matte smelting in the blast furnace.

Dr. Gahl has proposed limestone as a precipitant for the copper in the leaching solutions to avoid contamination of the water supply to the mill which would result from returning the exhausted liquors into the mill system. I think the difficulty could be overcome by applying the acid to a de-watered thickened pulp; this procedure would assist the leaching operation and the exhausted liquors could be run to waste.

Rudolf Gahl.—I would like to say that I agree completely with everything that Mr. Laist says, except, of course, his compliments. It occurred to me through Mr. Laist's remarks regarding the relative character of the froth produced by the different types of flotation machines, it might be worth while trying to modify the froth of the pneumatic machines by reducing the air supply, and perhaps also by reducing the frothing agent in the flotation oil mixture. I feel sure that the character of the froth can be modified to some extent, although I doubt very much whether without retreatment it ever would approximate the froth of the standard-type M. S. machine. Regarding Mr. Schimerka's remarks, I might say that I know very little about smelting, and have not considered the smelting problem very carefully. All I know is that Mr. Wallace, who used to be smelter superintendent of the International Smelting Co. here, assured me that he could smelt that stuff, and could pay for the lime also. Regarding the other point Mr. Schimerka brought out, about adding the acid to a thickened pulp, my impression is that still a large part of the water supply would be contaminated because we have to figure on leaching mainly very fine slime as it contains most of the oxidized copper; and it is the experience of every mill man that it is impossible to settle such slime to a consistency exceeding 3 to 1—that is, 3 parts of water to 1 part of solid matter. That would mean, if we added the acid to the thickened pulp, we would spoil 3 tons of water for each ton of leached ore.

C. E. Mills, Miami, Ariz.—It seems to me that one of the members might tell the experiences at Chino in leaching oxidized ore.

H. W. Morse, Los Angeles, Cal.—We have not gone far enough to say anything about what we are going to do, but I think at the next meeting of the Institute we will be able to give some very interesting results on this class of work. Dr. Ricketts has always turned down my proposals because they involved the handling of a thin pulp, settling and washing, and a great many other miseries which would certainly be involved in this class of work. But when he saw the proposed scheme at Chino, he expressed himself, fortunately, as being satisfied that we had, provided we could make it go, decided on a scheme that met his approval, which involved no thinning of material or washing, but simply the ordinary passage of mill pulp through an agitator and flotation machine. We have not any results worthy of consideration as yet, but I feel encouraged by the laboratory work and very confident of the future of this scheme. If it will go, it will be an important process because it will enable us to treat mixed ores, provided that they contain a sufficient amount of soluble ores to precipitate on iron, and provided that they contain sufficient sulphides to pay for flotation operation.

David Cole.—I understand that Dr. Ricketts "turned down" Dr. Morse when advocating leaching processes for low-grade tailings by methods which involved first getting the copper into solution and then separating the solution from the pulp in a clarified condition, which is difficult and involves much washing with clear water, etc., resulting in thin solutions, the latter entailing difficulties or "miseries" in effecting precipitation and recovery of the metal.

On the other hand, Dr. Ricketts approves the present process which Dr. Morse is using at Chino, in which the oxidized copper is first taken into solution by the use of about 3 lb. of sulphuric acid per pound of copper digested; this copper is then precipitated upon iron in an agitated mass, which results in metallizing the copper at the expense of about 2 lb. of iron per pound of copper precipitated. The metallized copper is then in a very fine state of division in the pulp, and the remaining copper sulphides which were not attacked by the acid are also in the pulp in a very fine state of division, and the whole is subjected to flotation treatment for the removal of both the sulphide and metallized contents, thus doing away with the necessity of removing the solution in a clarified condition from the pulp under treatment, avoiding the washing of the pulp for complete removal of solutions pregnant with copper, and avoiding most of the "miseries" previously unavoidable.

The exhibit of this new scheme of treatment operating on a small tonnage basis, which was so kindly thrown open to examination by our party while at Chino, was very interesting.

Miami has been experimenting for some time along the same line and has achieved success in a laboratory scale. Flotation has been successfully used in the separation of ultra-fine native copper in Michigan, and there seems to be much promise in this new scheme of treatment for the recovery of copper in low-grade ores when in a mixed condition of oxide and sulphide form. As Dr. Morse has suggested, the development promises to be important, and we may be on the threshold of another important step in copper metallurgy. I congratulate him.

RUDOLF GAHL.—Mr. Cole's question,* why hydraulic classifiers were installed in the Inspiration mill, is very pertinent, and his doubt about their usefulness for the classification of the deslimed flotation machine tailings seems well justified. In defense of the installation, I will say this:

1. The expense of operating these classifiers is very low, the principal cost item being labor for their attendance.

2. It requires a very small settling capacity to settle the table tailings. At the Inspiration mill it is accomplished by three 60-ft. Dorr settling tanks, with an additional tank installed for the purpose of keeping the coarsest sand out of the Dorr tanks. This small installation handles more than 7,000 tons per day. As the reclamation is accomplished right at the foot of the mill, the reclaimed water does not have to be lifted very high, and the reclaimed cost for this water is, therefore, low.

In other words, the benefit derived from the installation of hydraulic classifiers does not need to be very great to make them pay.

It is true that when we operated our test mill, we succeeded in obtaining quite satisfactory recoveries in spite of the fact that we put the whole of the flotation machine tailings on the tables without preliminary classification. We realized, however, that the table tailings (at least the coarser sizes) were not as low in their copper contents as we hoped to have them some day. We knew, furthermore, that by grinding finer we could reduce the coarser screen sizes in copper. As grinding finer than to a certain point is an economical impossibility, we had it in view to make different sizes and to send them to separate tables. This separation we intended to accomplish in hydraulic classifiers. The coarser tailings, or perhaps only middlings from the tables, could then be reground and reconcentrated.

To Mr. Cole's question as to what happens when the previously frothed sand feed is put upon the tables for final treatment without hydraulic sizing separation, I have to reply:

That we have not made the test which he suggests, but that I am inclined to agree with him that the tables might do as well on an unclassified as on a classified feed. The hydraulic classifiers were put in, as I

*See pp. 634 and 635, last paragraph.

have indicated before, in anticipation of further development of our flow sheet.

We are doing a great deal of experimental work on this special point, but have not gone far enough to predict just what the details of the additional sand treatment will be.

R. S. HANDY, Kellogg, Idaho.—I would like to ask if anyone has determined the relative economic efficiency of flotation as compared with gravity treatment on granular material.

C. E. MILLS.—I don't know whether I could answer that, but would say that through some experiments, which Dr. Gahl has mentioned, on the coarse feed to the tables, we have come to the conclusion that we do slightly better by flotation than by tabling of that material.

E. P. MATHEWSON, Anaconda, Mont.—I would like to say, in regard to Mr. Handy's question, that it is the practice in Montana to take out everything that is possible by means of tables or other water concentration machines; and at the Anaconda plant we take out concentrates 1½ in. size, and keep on taking out finer and finer material by water concentration until we get to the tables. What is left from the tables and not saved is then ground up and put through the flotation machines. We find by our experiments that it is better to keep the slime separated from the sand.

R. S. HANDY.—In experimenting on lead ore, I found it advisable to separate the granular material, tabling the sand material; and I wondered if that same experience had been gone through in copper ore.

C. E. MILLS.—That is similar to what we are doing at Inspiration in our Callow sections. The pulp is first subjected to a primary flotation, then sent to drag belts which separate slime from sand. The sand is tabled while the slime is given another flotation treatment.

DAVID COLE.—I consider it unnecessary to first remove the slime, for the reason that after the slime has been subjected to flotation treatment there is nothing in it that a table treatment can save—no sulphides in a sufficiently fine state of division to be transported by the slime, because the previous frothing operation has removed it all, and when slime is "denatured" in this manner it is no longer harmful upon the table, and does not interfere with the working of the sands upon the table. The table will do exactly the same work upon the sands that it would do if the slimes were not going across and off in the rear of the sands with the excess water. If no previous division or washing out of slime is practiced we have gained to the extent of the trouble and cost that would be entailed in making the division. That is the way I view the matter.

RUDOLF GAHL.—I would like to express a doubt as to Mr. Cole's

statement. Our experiences distinctly show that we get a better sand recovery by flotation if we remove the slime beforehand.*

E. P. Mathewson.—I beg to confirm Dr. Gahl's statement as to the practice at Anaconda.

R. S. Handy.—I would like to say that I have separated a minus 200-mesh lead ore into granular and flocculent matter, treating the former on tables and the latter by flotation. By this treatment I have recovered 92 per cent. of the total lead, in a concentrate assaying 63 per cent. lead. By flotation of the total ore I could recover only 85 per cent. of the lead when the concentrates assayed 63 per cent. lead and in order to get 92 per cent. extraction, the best grade of concentrates I could get was 50 per cent. lead.

L. D. Ricketts, New York, N. Y.—I think it has not been brought out that the ratio of concentration has something to do with the process in combined flotation and gravity concentration. At Anaconda, where they have a very low ratio of concentration, gravity concentration comes first, as it properly should, and flotation is applied only as a final cleaning-up process, directly to the slimes and to the sands after they have been ground sufficiently fine.

At Inspiration where the ratio of concentration is about 25 into 1 flotation is the primary process. All of the ore is ground before treatment to its ultimate fineness. Flotation is immediately applied. The coarser sands are then separated from the very fine sands and slimes and submitted to concentration and only a small portion of the copper is recovered by this means.

The point where flotation takes precedence over gravity concentration is, of course, not well defined. Gravity concentrates are more desirable for smelting purposes than flotation concentrates, and this should be kept in mind in association with the ratio of concentration and as much gravity concentrate should be obtained as practicable, providing the saving is not sacrificed.

A number of our mines have splendid concentrating plants, built before the days of flotation. In these, even if the ratio of concentration is high, gravity concentration may precede flotation for obvious reasons, but it must be kept in mind that if one could grind his ore to the ultimate desired fineness in one operation it would undoubtedly give a lower milling cost.

C. E. Mills.—I wish to say, in response to Mr. Cole's question, that I think Dr. Gahl overlooked something that we have at the Inspiration

*Note by Dr. Gahl: I judge from Mr. Cole's remarks in the form in which they appear on the stenographer's transcript of the discussion after being corrected by Mr. Cole that very likely he did not refer, as I understood him to do in the discussion, to the flotation treatment of a mixture of slime and sand. If this is so, my answer does not fit the case.

mill—a section that is running tables on an unclassified feed. The results from that section are just as good as in the others. I think that may be due to the fact that we do not do very good classification.

F. G. Cottrell, San Francisco, Cal.—There is one point in connection with what Mr. Handy was saying. I think what he has in mind is the distinction between sands under 200-mesh. It is more of a physical distinction than a mill distinction. I mention it because it might be something for the practical mill man in drawing a distinction between clay material and the very finely ground crystalline material. It is customary to lump under slimes everything, irrespective of its chemical properties.

C. E. Mills.—I think Mr. Cole has done a lot of work along that line. He might give us some information about it.

David Cole.—In my paper, The Advent of Flotation in the Clifton-Morenci District, I referred to that. In devising the early flow sheets for Morenci, we thought it would be advantageous to make the separation because we did not at the time consider flotation as at all applicable to the problem. In my paper I have alluded to the methods devised and reasons for making the divisions.

We knew, of course, that the separation could not be made upon screens. Hydraulic or washing separation I considered to be impracticable after trial of some of Mr. Overstrom's methods, and spitzkasten or pointed box or Callow tank separation is only partially successful. A better result would be accomplished by shortening the period of settling. To go into an explanation of why this is so would take too much time, but those who have been dealing with slime and know its peculiarities in settling and coagulation—its power to arrest the falling of successively larger particles of the ultra-fine sands which Mr. Handy has in mind—will realize what I mean by hastening the removal of the sands, and we developed a machine which was probably misnamed a "colloid separator," the office of which was to remove the mill man's type of "colloid"—the very much diluted clay—before it should have time to settle into a mass that would seize and hold the granular material which we knew we *could* treat successfully on tables or vanners, and at the same time allow the dilute clay, which it did no good to treat by table or vanner, to escape to the tails. In the light of our present knowledge of flotation treatment, we realize that this overflow, the dilute clay material referred to, is the part best prepared for flotation treatment, and outside of the oxidized copper content would yield the lowest tails and the most complete recovery, and that it is not necessary to remove the ultra-fine sands for flotation treatment, for these are best removed by flotation also.

The colloid separator that was devised for making the separation consisted of wide thin belts dragging upon the bottom of a shallow basin (2 in. deep), across which the pulp was fed in a wide, thin, gently moving

stream. The belts covered nearly the whole of the bottom of the basin (2 in. apart) and therefore practically the whole floor of the basin was (figuratively speaking) moving toward the discharge, where it rose gently above the surface of the water, received a gentle washing from clear water sprays like the sprays used upon vanners, and discharged its load upon vanners for gravity treatment, where a good grade of concentrate and a low tail were made. I think the illustrations in my paper will make the matter clear.

I have noted Mr. Mathewson's remarks concerning the present Anaconda practice, and that he has the impression that they really are desliming the feed before treatment on tables. The Anaconda flow sheet shows such a separation and the Anaconda type of conical deslimer is installed with that end in view, but like Mr. Mills' admission of a few minutes ago as to Inspiration's poor classification practice, Anaconda doesn't do good classification. The feed to the tables is not deslimed as it was intended it should be, the reason being that there are not enough of the Anaconda classifiers to do the work put upon them, and since the only office of these classifiers is to prepare feed for the Butchart riffle treatment, and since these tables have no office but to impoverish the ore treated by them, the complete separation of the slime from the feed is of little consequence, for all reject from the tables is taken at once to the regrinding mills where the cleaning-up work is most thoroughly accomplished by the flotation process. If the tables were making a reject to tailing, the Anaconda classifiers would have to do their full duty in desliming the feed to them, because the slimes going across the Butchart tables would result in serious losses, but since it is immaterial whether the primarily made slimes reach flotation treatment over the top of the classifiers or through the spigot, the classifiers' inefficiency and the results as to the reject from the tables are tolerable and there does not seem to be any reason to change. Obviously, it does not matter at Anaconda where the copper is taken out so long as a minimum amount of it is allowed to get away with the final tailing, and it is also obvious that with Anaconda's present practice, wherein the ore is reduced by their splendid treatment scheme from a 60-lb. copper content to less than a 3-lb. copper content in the final tail on a ratio of practically 3 into 1, the chance for improvement in mill practice through modification or more perfect slime classification is extremely remote.

Guy H. Ruggles, Miami, Ariz.—To get back to the subject of classification at the Inspiration: In the early part of the year tests were made showing that the section which treats the pulp from the flotation machines directly upon the tables, that is, without classification, did as good work as the other sections in which the pulp was classified. Since that time we have improved the settling in our drag classifiers The average of the general tailings on a number of sections for the month of July and

August shows that the section which treats the flotation pulp directly upon the tables is, if anything, not quite as good as the sections in which we have classification. There are some sections which beat this section by 0.02 per cent. copper and others which beat it by only 0.01 per cent. copper. Every 0.02 per cent. which we reduce the general tailings means 180 lb. of copper per 24 hr. per section, and when this is multiplied by 18 or 20 it means quite a lot. While our classification is not very good, it has helped out quite a little and if improved we might get more out of it.

R. C. Canby, Wallingford, Conn. (communication to the Secretary*).—At the time I was working upon flotation with the late Robert S. Towne, he told me of his interesting experience in the use of iron balls in a sort of ball mill, with iron balls used simply as an oiling device in connection with the Murex process.

Now in the Murex process there is no question of modification of surface tension, etc., simply the oiling by preferential affinity of the sulphide surfaces that the pulverized magnetic oxide may be made to adhere to the sulphides and not to the gangue.

Mr. Towne believed that the iron balls readily became coated with the oil and that *then* the sulphide surfaces very readily became coated as the particles—through the ball-mill action—came into contact with the surfaces of the oil-coated iron balls.

May it be that metallic iron has a somewhat similar assisting function in the cases referred to by Dr. Gahl? If so, it would suggest that an oil film upon the sulphide surfaces may be an assistance, if not altogether a necessity, in the frothing flotation process.

I had not intended to give Dr. Gahl the impression, as expressed in the footnote on page 586, that the experiments which I had conducted had given to Mr. Towne the idea of the porous medium for disseminating air bubbles. It was furthest from my thought to make such a claim.

What I suggested was the use of a Frenier pump spiral for incorporating the air bubbles, having in mind the converse of the Elmore vacuum, in which I proposed to produce the bubbles by pressure instead of by a vacuum. Upon investigation, however, I found that Norris had first suggested using air bubbles in solution.

While the manner of releasing the discharge from the Frenier into a separating compartment gave possibly an effect somewhat that of the "bubble column," still it was not actually the "bubble column" in the sense the word is now understood, and I did not intend to claim it as such.

I wish I felt it proper to take the space here to more fully attest the remarkably analytical manner in which Mr. Towne studied into all of the problems which he encountered, taking, as he always did, the lead in the constructive thought of his staff, with whom he worked in close touch.

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* Received Sept. 14, 1916.

Some Miscellaneous Wood Oils for Flotation.*

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(Arizona Meeting, September, 1916)

THE testing of flotation oils has occupied a large part of the time of the testing departments of various companies using the flotation process in the beneficiation of their ores. The great differences in ores and in oils has made such work necessary, as it has not been possible to select the oil best suited for the purpose without trial of many kinds. In fact, individual shipments of flotation oils from the same manufacturer will often differ so radically that the mill men have been forced to test every lot of flotation oil before its use in the mills.

Further, although almost every imaginable product of the distillation of wood, from alcohol to pitch, has been tested in a more or less perfunctory manner by various people, few companies have allowed the results of their tests to be made public.

This work was undertaken for the purpose of obtaining reliable data regarding the value as flotation agents of a number of products that are obtained from the distillation of wood by different methods, and incidentally to determine whether some of the products not used at present could be used. Large quantities of such materials are being wasted or burned in the wood-distillation plants of the United States, and if some use other than fuel could be found for these materials, such a use would represent a distinct saving to the country as a whole, would increase the profit of wood-distilling plants now operating and perhaps make it possible for others to operate which at present cannot do so except at a loss. At the same time it was thought that if such products can be used as flotation agents, then their use would serve to increase the number of oils and other products available for flotation purposes. It was also hoped that it might be possible to discover a substitute for some of the most effective but comparatively high-priced oils which are now used for flotation

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* By permission of the Director, U. S. Bureau of Mines.

This work is the result of coöperation between the Forest Products Laboratory of the Department of Agriculture, the Metallurgical Research Department of the University of Utah, and the U. S. Bureau of Mines. The authors are members of the three organizations.

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purposes. Hence these fractions occupy the major portion of the following list.

In taking up this work, the Forest Products Laboratory of the Department of Agriculture, which is interested in finding uses for byproducts from wood distillation, was represented by R. C. Palmer. The Bureau, which is interested in finding supplies of suitable flotation oils, was represented by O. C. Ralston, and the Department of Metallurgical Research of the University of Utah, which is doing coöperative work with the Bureau of Mines, was represented by G. L. Allen.

Three ores were chosen for tests of the series of oil samples herein described.

Ore A was from the mine of the Arthur Zinc Mining Co. near Ruby Valley, Nev. It was chosen on account of its simplicity. It consisted mainly of galena, sphalerite and quartz minerals, which were crystallized in such a way that they could be broken away from each other by crushing to the size necessary for flotation. Ground quartz was mixed with it to make it sufficiently low-grade. Any marked tendency of one mineral to float in preference to another could be observed with such a "free" ore. Consequently, the value of the oil for differential flotation of mixtures of galena and sphalerite could be observed; also the conditions which allowed the best extraction or the cleanest concentrate of any of these minerals with any given oil.

Ore B was the heavier sulphide ore from Cripple Creek, Colo. This siliceous ore contains some pyrite, some gold and silver as the tellurides, sylvanite and calaverite, and other tellurium minerals. The tests on this ore would allow observations on the flotation of pyrite as well as of the famous Cripple Creek gold telluride minerals. Wherever the pyrite contains gold it would be desirable to float it, while barren pyrite ought to be kept with the gangue. Hence the tests should be carried on with an ore containing both types of minerals, as was done in this case. This sample was furnished by the Golden Cycle Mining & Milling Co.

The ore C was the Utah Copper Co.'s milling ore from the mine at Bingham. A great many copper minerals are present in this ore—chalcopyrite, cupriferous pyrite, etc. The gangue is siliceous, and the ore is well known. It is a hard ore to treat by flotation and nearly any oil that will treat it is almost sure of success with most other copper ores. One peculiarity of the Utah Copper ore is that it is much easier to float in an alkaline solution.

Each of the ores was tested with three different tests—an acid test, an alkaline test, and a test run in neutral water. The water used in all the flotation tests was the Salt Lake City water. It contains a small amount of lime so that the water used in the "neutral" test was in fact slightly alkaline. This method of testing was adopted for the reason that it is known that some oils will refuse to work at all in neutral solution

although they do good work in acid solution. Others are more successful in alkaline solutions. The most advantageous amounts of acid or of sodium carbonate were not determined. The procedure followed, in order to facilitate work, was to add the reagent drop by drop to the charge in the flotation machine until it could be seen that it was causing some difference in the appearance of the froth or of the gangue. Further additions of the reagent were often made after most of the froth resulting had been skimmed off, and it frequently happened that a further amount of excellent froth would result.

Although such a method of obtaining the best conditions is not satisfying to certain types of mind, and is very aptly characterized as the "American" type of research, it is practically necessary to follow such a method in order to get anywhere with such testing. So many other variables are involved that to painstakingly follow out each one for only one oil, while the others are kept constant, is an immense undertaking. A flotation testing man must be an opportunist and seize on the best conditions for flotation as he is able to see them in his machine when varying the conditions, such as amount of agitation, depth of pulp in machine, amount of oil, amount of acid or alkali, temperature, etc.

The Janney test machine described in the *Mining and Scientific Press* for Jan. 1, 1916, was used in all this work. Our reason for using this machine is that it permits of quick work and is so designed that practically every particle of flotation concentrate and tailing can be recovered and weighed, while many other machines have inaccessible corners where some of the material escapes.

The extractions were calculated by the following well-known formula which does not require weights of the products involved.

$$\text{Extraction} = \frac{100\,(h - t)c}{h(c - t)}$$

where the analyses of the head, or original ore, tailing, and concentrate are represented by the letters in the equation. This formula assumed that any middling produced can be separated into concentrate and tailing.

Each test was "roughed" and "cleaned." The rough tail was final but the concentrate was poured back into the machine with more water and a clean concentrate and middling produced.

The oils tested in these experiments were furnished by the Forest Products Laboratory,[1] Madison, Wis. Most of them were either commercial samples or were produced in the laboratory in semi-commercial apparatus under conditions entirely comparable with plant practice.

The following is a somewhat detailed description of the different oils.

No. 23 was an authentic commercial pine oil, specific gravity 0.9385

[1] Maintained by the Forest Service, U. S. Department of Agriculture in cooperation with the University of Wisconsin.

at 21° C., with a sulphonation residue of 0.8 per cent. The oil was produced by the steam distillation of fat southern pine wood, the crude oil being redistilled to separate the turpentine (boiling point 155 to 170° C.), from the pine oil (boiling point 190 to 220° C.). As there is some confusion in the mining industry in regard to the name "pine oil," many other oils produced from dead pine wood also being known as "pine oil," it may be well to describe the oil to which the name is correctly applied. There is a resin in the sapwood of the living pine, which flows out when the tree is wounded. This resin is the source of the ordinary turpentine and rosin of commerce. When the sapwood turns to heartwood, the resin becomes hard and during the aging process another oil is formed which did not exist in the sapwood. This oil is heavier than turpentine, has an entirely different composition, and a much higher boiling point, and it is this oil that is correctly termed "pine oil." This oil can be removed from the dead pine wood either by the action of steam or dry heat. Dry heat carefully controlled so as not to exceed the maximum boiling point of a the oil will remove it in practically as pure condition as when the wood is treated with saturated steam. Pine oil is produced either as a yellow or water-white oil, the former being produced in the first separation of the crude turpentine-pine-oil mixture, and the latter by redistillation of the yellow oil. If the wood is heated at a much higher temperature, as it is when the temperature is not controlled during the early stages of the destructive distillation process, the natural oils (turpentine and pine oil) in the dead wood are contaminated by tar and tar oils produced by the decomposition of rosin and wood. It is practically impossible commercially to separate a pure pine oil from this mixture.

Chemically, pine oil has been found to be composed chiefly of a terpene called terpeneol, although other terpenes such as borneol, fenchyl alcohol, and cineol, are also present. Since the detection of mineral oils as adulterants in pine oil is of special interest it may be well to point out that the sulphonation test may be successfully used for the detection of mineral oil. A pure pine oil should give a sulphonation residue of less than 1 per cent. Special care must be taken, however, in carrying out the test, and it is suggested that the test be made as follows:

Take 25 c.c. of 37 normal sulphuric acid in a Babcock bottle and add 5 c.c. of the oil to be tested, 1 c.c. at a time, shaking vigorously after each addition. When the oil is all added, place the bottle in a water bath and heat at 100° for 1 hr., shaking cautiously from time to time to insure a perfect mixture of the oil and acid. Cool and fill the bottle with ordinary sulphuric acid and either allow it to stand for 24 hr. or centrifuge it in order to read the residue. It is absolutely essential to have the strength of the acid exactly 37 normal; the cooking of the mixture for 1 hr. is equally important for securing low residue with pure pine oil. The index of refraction of the residue is also of value. With pure pine oil the residue

should have an index of over 1.5, while a mineral oil will have a residue with an index of less than 1.48.

A pine oil was included in these tests for the purpose of comparing an oil of known flotation value with the other oils. The sample had been shaken repeatedly with hot water in order to remove any water-soluble constituents (probably the fenchyl alcohol), to determine whether the removal of water-soluble material had any noticeable effect on the value of the oil for flotation. About 5 per cent. was found to be soluble in water.

No. 11 was a commercial wood oil produced in the destructive distillation of hardwood, the particular species in this case being maple and birch. The specific gravity of the oil was 0.934 at 21° C. The chemical composition of this oil is unknown, although it probably contains a number of complex ketones. In the destructive distillation of hardwood a heavy tar settles out of the crude pyroligneous acid, the watery distillate containing the wood alcohol and acetic acid, and this tar has included in it some of the pyroligneous acid. To remove this acid the tar is usually placed in a wooden still and either washed by blowing in exhaust steam or is simply heated with steam in closed coils. In either case a watery product distills over and with it is carried this light tar oil or "wood oil," as it is called in the plant. This oil is practically a waste product and is burned for fuel in most plants. It is an excellent solvent, however, and a small amount is sold for that purpose. A cord of hardwood gives on an average about 2 gal. of this oil and the production in all the hardwood distillation plants is probably close to 2,500,000 gal. annually. It could probably be bought for $0.10 to $0.15 per gallon f.o.b. plant, as this is the price usually paid for it in the crude condition for a solvent.

No. 18 was a crude hardwood tar just as it settled out of the pyroligneous acid, as described above. This tar was produced in a semi-commercial laboratory distillation plant by the distillation of equal proportions of beech, birch and maple. It still contained the pyroligneous acid and wood oil. On distillation the sample gave 12 per cent. pyroligneous acid, 8 per cent. wood oil, 42 per cent. wood creosote oil, and 38 per cent. wood pitch. The specific gravity of the combined wood oil and wood creosote was 1.067 at 21° C. When the distillation of the crude tar is continued with direct heat after distilling off the watery distillate and light oil (wood oil) the oil obtained is similar to the hardwood creosote now on the market for flotation. The object in testing a crude hardwood tar of this kind was primarily to determine to what extent the pitch present in the tar acted as a deleterious adulterant. Excluding the wood oil, about 15 gal. of this tar is produced per cord of hardwood, making available per year over 20,000,000 gal. of this crude tar. This tar with the wood oil and pyroligneous acid removed is generally used for fuel at the distillation plant, where it is worth about $0.03 per gallon for this purpose,

in comparison with the normal prices for other kinds of fuel. It could probably be bought for \$0.05 to \$0.06 per gallon f.o.b. plant.

No. 15 was an entirely different tar product produced in the distillation of hardwood. After the pyroligneous acid has been allowed to stand to settle out the oily tar, it contains a tar-like substance dissolved in it which has to be removed before the liquor can be worked up into alcohol and acetic acid. Since this tar is non-volatile it is removed by completely distilling the pyroligneous acid, the tar being left as a residue in the still. This sample was produced in the laboratory distillation plant from beech, birch and maple. It is a heavy, non-volatile, viscous, tar-like substance, specific gravity about 1.4. Its composition is unknown. It is about 50 per cent. soluble in water, the remainder dissolving in the pyroligneous acid because of the solvent action of the alcohol and acid constituents. The yield of this "dissolved tar" or "residual tar" or "acid water tar" or "copper still tar," which are the various names given to it in the plants, is from 6 to 8 gal. per cord, making a total production in the United States of about 9,000,000 gal. per year. It is strictly a waste product at present, although worth about \$0.025 per gallon as fuel. It could probably be obtained for \$0.04 to \$0.05 per gallon at the plant.

No. 16 is a 15 per cent. water solution of the water-soluble portion of No. 15, just described. The specific gravity of the solution was 1.061 at 21° C. This sample was prepared to determine the value of the water-soluble portion as compared to the whole tar.

There have been many references in the literature to the use of pyroligneous acid for flotation, but no data have been given showing the actual value of this material. These tests included several samples of pyroligneous acid to determine this point.

No. 14 was a semi-commercial pyroligneous acid produced by distilling maple. The specific gravity of the sample was 1.036 at 21° C. It contained 9.7 per cent. dissolved tar, 4.8 per cent. wood alcohol, 11.9 per cent. acetic acid. The value of this liquor for its alcohol and acid content is over \$0.05 per gallon.

No. 12 was a commercial pyroligneous acid from a plant distilling birch and maple, from which the dissolved tar had been removed by distillation. The specific gravity of the sample was 1.008 at 21° C. It contained 3.1 per cent. wood alcohol and 5.9 per cent. acetic acid, making it worth a little over \$0.03 per gallon for its content of these products.

No. 10 was commercial crude pyroligneous acid produced in the destructive distillation of southern yellow pine. The specific gravity of the sample was 1.034 at 21° C. It contained much less alcohol and acetic acid than the hardwood pyroligneous acids, the composition being 1.8 per cent. alcohol, 2.85 per cent. acid and 7 per cent. dissolved tar. Except under abnormal conditions it does not pay to refine pine-wood pyroligneous acid in competition with the hardwood product and it is thrown to

waste. This particular sample would not be worth over $0.0175 per gallon for the alcohol and acid in it. A cord of pine (4,000 lb.) will yield from 125 to 150 gal. of crude pyroligneous acid. It would certainly not pay to buy the crude acid at even $0.02 per gallon f.o.b. plant on account of the freight charges on the large volume of water in it. The concentration of the crude distillate would simply give a thick tar containing a small amount of the acid and none of the alcohol, but if the tar is the valuable constituent this would seem to be the only feasible method of marketing softwood pyroligneous acid for flotation.

The Results

The results of the flotation of these three ores with the above set of oil samples are contained in Tables 1 to 3. In Table 1 the analyses of the concentrate and tailing for each test are given. In the other two tables only the analyses of the concentrate samples are given. Accompanying the analyses of the concentrate samples in each case are the percentages of extraction obtained. The test in neutral, acid and alkaline solutions for each oil is given.

It can be seen that many of the oils give fully as satisfactory results as pine oil for practically all the minerals tested. Oils 11, 18, and 15 are the ones of most interest on account of being immediately available at low prices and because they can be used in small amounts with success. For all-round work, No. 18 seems to have given good extractions of all the metals tested, together with fair grades of concentrate. It will be remembered that this is the crude hardwood tar which has settled out of the pyroligneous acid.

It is a pleasant surprise to find that No. 16, the 15 per cent. solution of No. 15 in water, is so efficient in comparison with No. 15. Here is something which is really not an oil because it is completely soluble in water. It is satisfactory for all the minerals except the copper minerals of ore C.

It will be noticed that the best extractions of the zinc, and generally the highest-grade zinc concentrates, were obtained in solutions that had been acidified. Differential flotation of galena in the presence of sphalerite is seen to be most marked in alkaline or in neutral solutions. No. 14, the hardwood pyroligneous acid, seems to be best adapted for this.

The extractions of the gold in the Cripple Creek ore and the grades of concentrate are surprisingly good. Oils 18, 15 and 16 seem to be capable of producing the highest-grade concentrates. By further treatment of the tailing it should be possible to get even higher extractions, although the grade of the total concentrate would be cut down. No attempts to run tests in this manner were made, as the object of the work was the finding of oils best adapted to do certain things. The relatively high

TABLE 1.—*Tests on Ore A in Neutral, Acid and Alkaline Solutions with Different Oils*

Analysis Head: Lead, 3 4 Per Cent ; Zinc, 4.74 Per Cent.

Oil No	Character of Oil	Remarks	Total Time, Min.	Total Oil, Lb. per Ton	Total Acid, Lb. per Ton	Total Alkali, Lb per Ton	Lead		Zinc	
							Per Cent.	Per Cent Extraction	Per Cent	Per Cent. Extraction
23	Pine oil..	Concentrate	10	0.375	....	...	55.9	53.1	12 4	62.2
		tailing.	10	0.375	....	...	1.6	. ..	2 3	
			10	0.375	11.14	...	32 3	87.9	37 9	78 2
			10	....	3 36	...	0 4	..	1 1	
			15	0.375	....	8 0	18.0	89.5	25.5	79.1
			10	.. .	....	2 0	0.4	..	4 4	
11	Hardwood "wood oil".	.. ..	17	1.11	....	...	57 2	92 6	20 4	58.2
			. 10	0 18	....	...	0.3	....	2 3	
			16	0.94	3 68	...	37 9	88.7	34 2	72 6
			15	0 17	3.68	..	0 4	. ..	1 4	
			15	1.29	....	4 0	48 0	97.2	32 0	64 8
			5	.	....	...	0 1	. ..	1.8	
18	Crude hardwood "settledtar."	....	17	1.35	....	...	48 6	97.2	28 0	68 4
			3	...	....	...	0.1	....	1 7	
			19	1.24	3.68	...	37.7	98 2	33 4	76.5
			6	....	....	...	0 1	....	1.2	
			20	1.70	....	2.0	48 7	98.0	27 9	63 4
			4	....	....	...	0.1	. ..	1 9	
15	"Dissolved hardwood tar"	. ... . .	20	1.44	....	...	59 2	79 4	16 2	30.2
			7	....	...	...	0 7	.	3 5	
			15	1 44	5 52	...	32 8	88 7	13.0	69 8
			10	0.72	3 68	...	0 4	..	1 9	
			20	1.44	....	4.0	66 7	73 0	38.1	30.3
			8	....	....	...	1 0	....	3 4	
16	Water solution of No. 15	.. ..	13	1.80	....	...	55.1	90 2	22.0	43.7
			5	....	....	...	0 4	....	2 9	
			19	1.80	7.36	...	30.3	92.2	40 5	77.2
			4	....	....	...	0.3	....	1 2	
			19	1.80	....	4.0	53 7	77 4	23 3	44 7
			4	....	....	...	0 8	....	2.9	
14	Crude hardwood pyroligneous acid.	....	20	14.00	....	...	54.3	86 2	22 7	51.5
			3	....	....	...	0 5	....	2.6	
			15	14.00	5.5	...	29.5	72 2	41 5	70 0
			5	....	. ..	...	1 0	. ..	1 5	
			17	12.0	....	4.0	62 0	54.8	9.8	42 7
			6	....	....	...	1.5	....	3.4	
12	Tar-free hardwood pyroligneous acid.	....	13	20.0	3.68	...	23 5	42 7	31 7	41.4
10	Crude softwood pyroligneous acid.	. ...	17	10.0	....	...	53 8	73 3	23.9	41.3
			13	4 75	..	. .	1 0	..	3.0	
			18	12.0	3.68	...	31 6	64 7	35.2	57.3
		No cleaner	7	4.00	.. .	.	1 3	....	2.2	
		test	12	16 00	.	4 0	49 1	57.8	17.9	15.0

grade of the heads in the tests on the Cripple Creek ore leaves tailing in all these tests too high to compare favorably with roasting and cyaniding. Hence, further work in adaptation of these oils to give higher extractions would have to be done. This is a question of mechanical manipulation with which every flotation experimenter is familiar. The extractions of the iron in the Cripple Creek ore are very poor with all the tests. As the ore was ground dry in a laboratory sample pulverizer this is not surprising, for pyrite is peculiarly sensitive to oxidation during such grinding and will not float well. Not being an important mineral in this case, the method of preparation of the sample did not matter. It is interesting to note that No. 11 and No. 10, the "wood oil" and the softwood crude pyroligneous acid, respectively, were the two oils which gave best extractions of the iron.

As was expected, the Utah Copper ore usually floated best in alkaline solutions, oils 11, 18 and 15 doing the best work.

Further work is being done, especially in the testing of the residues left by concentrating the softwood pyroligneous acid by evaporation, and many other oils are being tested in this manner.

TABLE 2.—*Tests on Ore B in Neutral, Acid and Alkaline Solutions with Different Oils*

Analysis Head: Gold, 0 608 Oz. per Ton; Iron, 5.2 Per Cent

Oil No	Total Time, Minutes	Total Oil, Lb per Ton	Total Acid, Lb per Ton	Total Alkali, Lb. per Ton	Gold*		Iron*	
					Oz per Ton	Per Cent. Extraction	Per Cent	Per Cent. Extraction
23	21	0 375	0.00	0.0	4.50	81.5	8.0	10.2
	21	0.575	3.68	0.0	6.42	78 7	9.0	16 5
	21	0.375	0.00	1.0	4.76	79 7	8.2	9.9
11	24	1.11	0.00	0.0	4.72	82.5	8.5	31.9
	22	1.11	3.68	0 0	6.12	75 8	17.2	41.1
	23	1.11	0.00	4 0	5.64	82.3	13.4	14.8
18	22	1.8	0 00	0 0	10 34	78 2	13.2	17.7
	21	1.5	3.68	0.0	11.52	78 0	14.3	19 6
	22	1 5	0.00	1.0	11.60	81.2	13.1	6.2
15	23	2.88	0.00	0.0	5.06	82.3	5.7	0.0
	21	2.88	3.68	0.0	13 80	74.8	9.6	11.8
	21	2.88	0.00	1.0	5 80	76.0	7.4	6.2
16	21	2 4	0.00	0 0	10.10	81.3	8.6	31.5
	21	2.4	3.68	0.0	12 60	81.2	...	0.0
	21	2 4	0.00	1.0	7.88	78.7	11.7	27.3
14	21	12 0	0.00	0.0	5.42	79.1	9.2	32.5
	22	12.0	3 68	0.0	7 22	81.7	12 0	40.3
	19	12.0	0.00	1.0	7.28	78 7	12 1	26.9
12	15	18.0	0.00	0 0	5.88	76.0	7.7	32.3
	20	20 0	3.68	0.0	6 76	81.7	15.0	12.8
	15	24.0	0.00	1.0	9.44	77.0	6.9	14.0
10	23	16 0	0.00	0.0	5.40	88.7	6.1	40.8
	21	16 0	3.68	0.0	4 98	79.6	6.0	41.2
	24	16.0	0.00	1.0	6 10	79 2	6.5	16.7

* In concentrate samples.

TABLE 3.—*Tests on Ore C in Neutral, Acid and Alkaline Solutions with Different Oils*

Heads: Copper, 1.58 Per Cent.

Oil No	Total Time, Minutes	Total Oil, Lb. per Ton	Total Acid, Lb per Ton	Total Alkali, Lb. per Ton	Copper*	
					Per Cent.	Per Cent Extraction
23	17 16 17	0 375 0 375 0.375	0 00 7 36 0.00	0 0 4	3 83 8.61 1.53	28.7
11	20 21 21	0 92 1.48 1 29	0 00 7 36 0 00	0 0 6	15.95 16.15	29.3 32 6
18	20 20 20	1.8 1 5 1.2	0 00 7 36 0 00	0 0 4	16 45 2.48 22.27	51 2 68.2
15	21 21 23	1 44 1 44 1 44	0 00 7 36 0.00	0 0 4	5.84 2.77 18.17	4 3 23.1
16	17 18 23	2 1 2 7 2 4	0 00 7.36 0 00	0 0 8	4.50 1 72 7 94	33.6 5.6 3.9
14	20 18 19	5 0 5.0 7 0	0 00 7 36 0 00	0 0 4	2.29 1.43 12 90	 13.5
10	19 19 12	8 0 10 0 8.0	0 00 7.36 0 00	0 0 7	1 24 1 43 2 29	 25.3

* In concentrate samples.

TABLE 4.—*Screen Analyses*

Ore A	Opening	Mesh	Weights	Assays	
	Mm.		Per Cent	Pb, Per Cent.	Zn, Per Cent.
Through.	0.589	28	100.00		
On.	0.417	35	0.43	0 00	0.00
On.	0 208	65	12.88	0 05	0 98
On.	0.147	100	34.46	0.07	1 37
On.	0 104	150	23 76	0 07	2.24
On.	0 074	200	10 59	2 19	6 46
Through.	0.074	200	17.75	19.08	14.60
Ore B					
Through.	....	200	100.00		
Ore C					
Through.	0.295	48	100 0		
On.	0 208	65	2 2	1.58	
On.	0.147	100	16.7	1 58	
On.	0 104	150	16.0	1.82	
On.	0 074	200	53.3	1.46	
Through.	0.074	200	11.8	1.73	

The Advent of Flotation in the Clifton-Morenci District, Arizona

BY DAVID COLE,* EL PASO, TEXAS

(Arizona Meeting, September, 1916)

At the time flotation appeared upon the metallurgical horizon in Arizona, the writer, under the direction of Dr. Ricketts, was engaged in remodeling and enlarging the No. 6 Concentration Plant of the Arizona Copper Co. at Morenci, and the work had been in progress nearly a year before the Inspiration experiments with flotation had disclosed the revolution in concentration that was at that time impending.

The simplified flow sheet worked out for the remodeling of the Morenci plant had been based upon the removal of the freed metal in a minimum number of stages by treatment upon tables equipped with Butchart riffles, the latter being adapted to accomplish both classification of feed and removal of the metal at one operation, substantially as described in my previous paper.[1]

The scheme of treatment for the slime was based upon the well-known fact that after copper sulphides, such as chalcocite and chalcopyrite, etc., are reduced to a certain extremely fine state of comminution they are definitely beyond the reach of separation upon any of the concentrating devices then known.

By using drag-belt classifiers (which served as conveyors as well as separators), the overflow would be of the usual "slimes" class. Experiment had shown that when these drag-belt overflows were properly diluted (that is, to about 5 to 7 per cent. solids in the feed under treatment) the very fine sand, and especially the very fine but still granular sulphide particles, would, if given a short distance to fall, quickly settle out in prime condition to yield an excellent recovery on vanners; a further valuable feature would be that the very fine non-separable final overflow tonnage might be discharged direct to tailing, thus conserving space requirements so as to permit within the old building, without embarrassment, the increase of capacity required.

For reasons that are obvious, this kind of feed preparation for the slime could not be successfully accomplished in any form of pointed boxes or

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* Consulting Engineer.

[1] Development of the Butchart Riffle System at Morenci, *Trans.*, vol. 51, p. 405 (1915).

spitzkasten device, therefore a further elaboration of the drag-belt idea was worked out as the best method for accomplishing the separation. This machine was known as a Colloid Separator and is shown in Figs. 1 and 2. It works on the premise that nearly all of what may be called

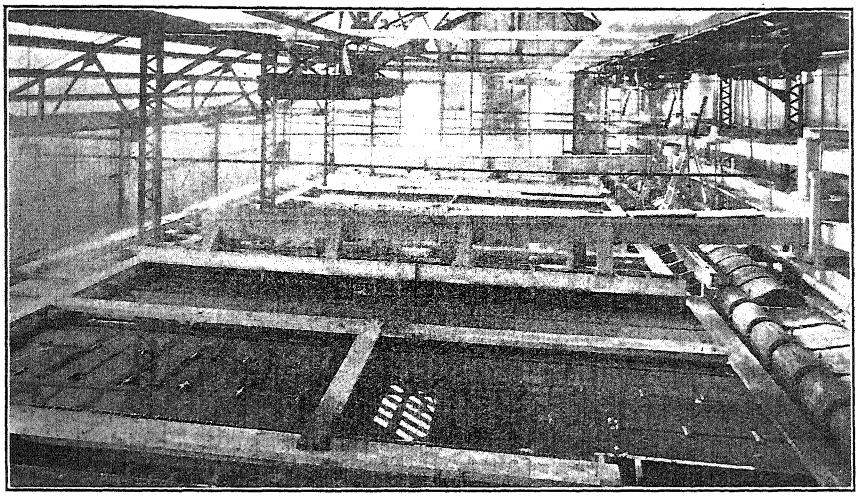

FIG. 1.—DRAG-BELT SEPARATORS IN MILL OF ARIZONA COPPER CO.

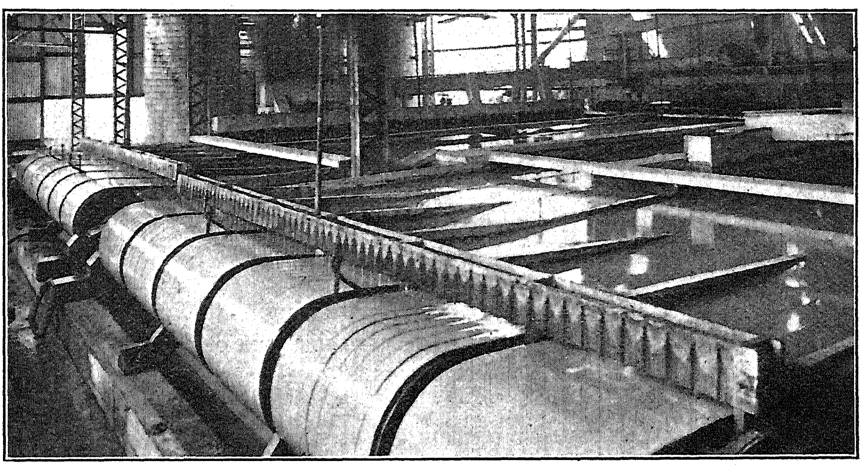

FIG. 2.—ANOTHER VIEW OF DRAG-BELT SEPARATORS.

ponderable material in the thinned pulp, falling but 2 in., will lodge upon the belts and be quickly removed, while the flocculent slime material will remain in suspension and go away with the overflow. In this way a feed is prepared for the vanners containing a maximum amount of the very fine but granular sulphides and a minimum of colloidal material; at the

same time the drag-belt overflow product contains a minimum of sulphide particles and a maximum of flocculent slime or colloidal pulp. This final overflow was found in practice to be approximately two-thirds of the total tonnage handled by the belts and treatment upon vanners was, as far as copper recovery was concerned, devoid of beneficial results. The copper escaping in this overflow was in the form of extremely fine chalcocite, bornite with a little pyrite and chalcopyrite, together with oxidized, and water-soluble, copper salts. Taken together, these gave the overflow a copper tenor of from 1 to 1.3 per cent. in a ton of dry material, thereby

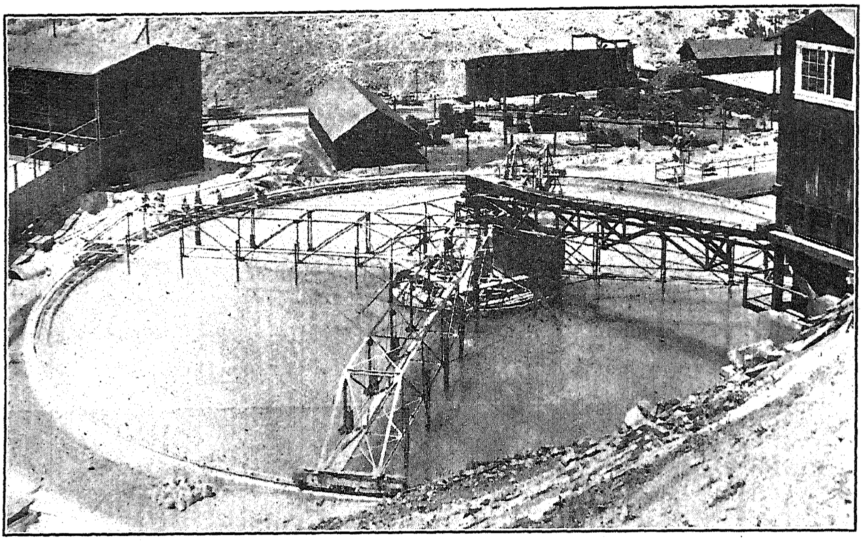

FIG. 3.—DORR THICKENER OF 130-FT. DIAMETER AT ARIZONA COPPER CO.'S PLANT.

accounting for the larger part of the tailing losses. However, this overflow, with its ultrafine and otherwise handicapped copper-bearing material, was practically beyond the reach of any concentrating machine at that time known, and could therefore go to tailing. A Dorr thickener[2] 130 ft. in diameter, the first one of such large size, was devised to recover the water from this overflow before it was allowed to go to waste. (See Fig. 3.)

By this plan, the treatable portion of the slime could be handled by the complement of vanners already installed in the company's mill. This arrangement, in conjunction with the saving in floor space, resulting from the introduction of the Butchart riffle, made it easily possible to double the capacity of the plant under practically the original roof. While it did not promise to recover a larger percentage than usual of the

[2] Described in *Engineering and Mining Journal*, Vol. 100, No. 4, p. 131 (July 24, 1915).

truly slimed copper, it did promise to give the best possible results in one-half the space.

By the time this flow sheet had been put into practical operation, the Inspiration experiments were attracting widespread attention and flotation was beginning to be taken seriously as a concentration agent for the handling of copper ores. It was, however, still regarded as an auxiliary process and was thought to be inapplicable to ores carrying an excess of talc or clay like the Clifton-Morenci ores. Some experiments with the Elmore process on these ores in former years had been unsuccessful and laboratory work which we did later on a very small scale with the meager information available seemed to corroborate this theory. Space was reserved, however, in a proper place in the mill building, for the installation of flotation equipment in case further development should prove its adaptability—at least for some appreciable portion of the tonnage.

Meanwhile, the Inspiration company had built the 600-ton "pilot" mill in which the new process was rapidly graduating from an auxiliary into the main method of separation, in the manner so fully and interestingly described by Dr. Gahl.[3] Starting with semi-mysterious compounds, Inspiration had soon found that simple flotation reagents were equally efficacious. Through the kindness of Mr. Mills, I secured a drum of cresylic acid and some pine oil with which to try a few experiments at Morenci.

The tailing from the No. 6 Concentrator was at that time discharged into Morenci Canyon and cascaded along for about 1 mile before being taken into a flume to be carried to the impounding dams. The creek bed was rough and steep, inducing great agitation of the pulp and resulting in the production of large amounts of white froth which floated down the stream. This froth carried no concentrations of copper minerals, but I thought it might be possible to change its character and produce a mineral froth by the use of flotation reagents introduced where the tailing left the mill, and thus possibly secure from the natural situation afforded some benefit at little cost.

A small can of the cresylic acid was arranged to drip into the tailings launder at a point where the tailings made the initial plunge into the creek bed. The results were instantaneous and very gratifying. Black froth began to collect in eddies and float downstream for a few yards to a second plunge where we were greatly surprised to find that it became white again on account of the instant dropping of the metallic load. Feeding the reagent into the stream immediately above the second plunge would not cause a mineral froth to rise as in the first plunge, and the reason was finally located as being the effect of a town sewer which was discharging under the surface into the creek between the two pools; the sewage effectively killed the metal-carrying capacity of the froth.

[3] *History of the Flotation Process at Inspiration*, p. 576.

Cresylic acid was then added to the feed of a regrinding Hardinge mill which was discharging into a long drag-belt classifier. The results were again most encouraging; black mineral froth began immediately to appear and to collect in large volume upon the relatively still water in the drag-belt trough. This rough froth concentrate was found to assay over 35 per cent. copper; the product contained 35 per cent. insolubles, mostly in the form of coarse sand, mechanically suspended in the froth and easily separated by screening. It was found that 1 per cent. of the copper in the froth concentrate was in oxidized form; 22 per cent. of the concentrate was too coarse to pass a 100-mesh screen, and this portion carried only 0.87 per cent. copper, while the minus 100-mesh material carried 46 per cent. copper, 1.32 per cent. of which was oxidized, and but 16 per cent. of insolubles. The high grade of the froth concentrate was astonishing, showing that chalcocite predominated in it.

A few days later I made a bank of tube grates, consisting of six parallel 1-in. pipes made up with return bends. The pipes were drilled full of small holes and were wrapped with cotton blanket tied with spirally wound wire. This tube-grate air filter was put into the drag-belt trough as deeply as possible, without touching the belt. Coarse sand could pass through between the grates and be removed by the belt underneath. The pipes were supplied with compressed air for the purpose of creating additional froth, and it was found that the product made, without further treatment, assayed 40 per cent. copper, 1.14 per cent. of which was oxidized, and that it carried but 20.4 per cent. insolubles.

Plans for a small frothing machine of the mechanical-agitation type were immediately made, and on July 20 the apparatus was tried with the colloid separator fines as feed, with the following remarkable result: Feed, 2.32 per cent. copper, of which 0.62 per cent. was oxidized; the concentrates produced assayed 20.4 per cent. copper, of which 1.18 per cent. was oxidized; the tailing carried 0.52 per cent. copper, of which 0.38 per cent. was oxidized, leaving only 0.14 per cent. sulphide copper as the rejection of the machine. This was an extraction of more than 79 per cent. of the total copper, and more than 92 per cent. of the available (sulphide) copper. This showed clearly that much could be expected in the application of the new process to Morenci ores.

The Cananea Consolidated Copper Co., in Sonora, Mexico, had been experimenting with the use of some Flinn-Towne pneumatic flotation units in its concentrating department. The plant had been temporarily shut down on account of revolutionary troubles, and arrangements were made by Dr. Ricketts for the removal of one of these units to the No. 6 Concentrator at Morenci. The apparatus was installed under the direction of the Flinn-Towne people in the space reserved for flotation and was operated for several weeks with very gratifying results as to recoveries. These experiments demonstrated clearly that the flotation process

would be suited to the recovery of slimed copper sulphides in the Morenci ores. But the Flinn-Towne units were not of a size suitable for use in the equipment of a large plant, or for the handling of large tonnages, except by using a great number of them. It was thought that their capacity could not be enlarged to advantage because of the difficulty with the air-emitting medium used, which was in circular-disk form with central discharge. These disks could not be made larger in diameter

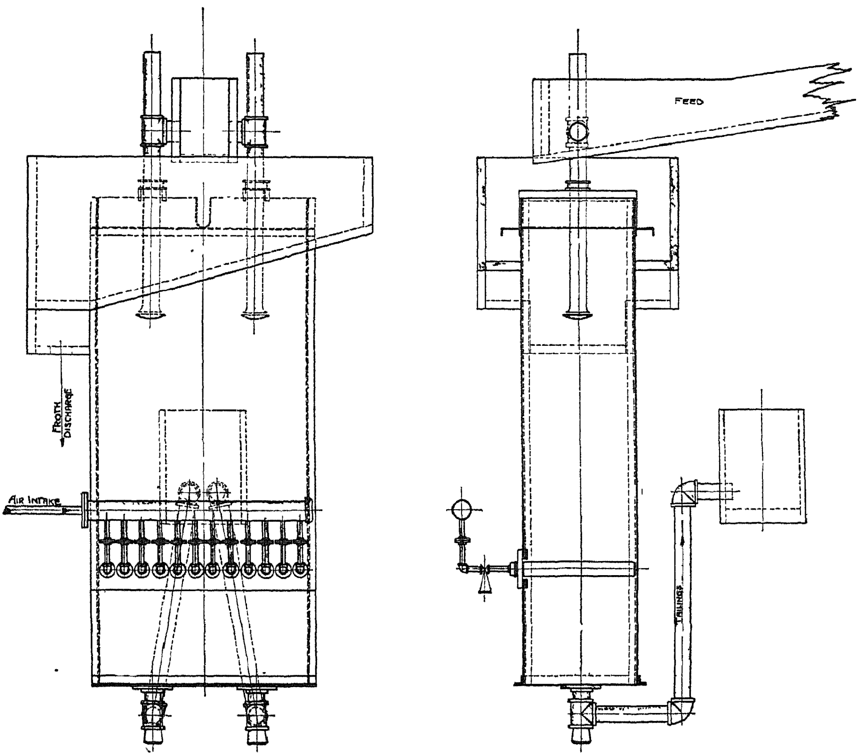

Fig. 4.—Simple Tube-Grate Cell.

without increasing the difficulty coming from "blinding" of the air-emitting surfaces, through the lodging of coarse particles upon them, and from the formation of vortices by the larger volumes discharged through the single opening in the center which entrained air bubbles with the reject.

The tube-grate idea previously tried in the drag-belt tank seemed to be a better way to admit the air because nothing could lodge upon the air-emitting elements to blind them, and constriction of the passage for the pulp and water would be avoided. This tube-grate idea therefore formed a basis on which to design units of large capacity for practical

mill work and especially to solve the limited-space problem at No. 6 Concentrator. Accordingly, a tube-grate cell was installed, in January, 1915. This simple cell is shown in Fig. 4. It was used for some time to demonstrate the tube-grate idea and served as a "cleaner" in the subsequent work done with a full-sized machine.

The demonstration of the new tube-grate cell was such a success that a three-stage machine, to have a capacity of 400 tons per day, called the "C-B" machine, was designed and made. Another one of the same kind and size was made concurrently for the Inspiration Consolidated Copper Co., and both of them were started in operation early in March. The Inspiration machine, which is illustrated in Dr. Gahl's paper, gave good results, proving the design to be substantially correct. The Morenci machine was working in corrosive water, which resulted in the formation of rust on the steel tubes and the gradual closing of the openings. The air supply was found to be contaminated with grease and oil from the blower bearings, entrained muddy water in the air, etc., which closed the pores of the air filter from the inside. Some delay was experienced in overcoming these difficulties. That they were of minor importance was proved by the almost uninterrupted success and approximately perfect operation of the duplicate machine running concurrently at Inspiration.

The C-B machine at Morenci made large volumes of very rich froth and had become immediately profitable by reason of its being able to handle a large tonnage and save copper which would be otherwise beyond the reach of concentration. It was therefore kept in operation as a profit maker, even though working under the handicap of partially clogged tubes, blower troubles, etc. The daily tonnage handled during the month of April, 1915, was from 125 to 390 tons per day, with an average of 209; the recovery made by the machine was from 35 to 79 per cent. of the sulphide copper present in the feed, with an average for the period of 65 per cent.

The blower used was an old one borrowed from the mining department, where it had been used in ventilation. It was designed for not more than 3½ lb. pressure, and the developing of 6 lb. in it deflected the shafts and caused the impellers to rub upon the sides of the machine, which had to be water-jacketed to keep down the heat developed. It was much larger than necessary, and a great excess of air was blown off from open valves. Therefore, no record of the amount of air used or power required, could be even approximately obtained.

In spite of these minor difficulties it was proved: That flotation could be applied to these ores with great advantage; that the copper in the mill tailing could be reduced to 0.50 per cent. (of which 0.25 to 0.30 per cent. was oxidized and beyond the reach even of flotation); that this result could be improved by finer grinding in the Hardinge mills; that the

TABLE 1.—*Detailed Results of Tests on C-B and Callow Flotation Machines for Month of July*, 1915.

SCREEN ANALYSIS, C-B MACHINE

Machine Tails

Mesh	Solids: Per Cent Wt.	Solids: Cum. Per Cent. Wt.	Assay: Per Cent Total Cu	Assay: Per Cent Oxidized Cu	Per Cent. Wt × Per Cent Cu	Copper: Per Cent. of Total	Copper: Cum Per Cent. of Total
+ 20							
+ 28	1 1	1 1	0 77		0 85	1 41	1.41
+ 35	3 8	4 9	0 65		2 47	5 51	5 51
+ 48	8 0	12 9	0 79		6 32	10 49	16 00
+ 65	9 9	22 8	0 80		7 92	13 14	29 14
+100	14 7	37 5	0 87		12.79	21.23	50.37
+150	11 2	48 7	0 79		8 85	14 69	65 06
+200	6 9	55 6	0 67		4.62	7.67	72.73
−200	44.4	100 0	0 37		16 43	27 27	100 00
Total	60 0				60 25	100.00	
Check			0 61	0 24			

Machine Concentrates

Mesh	Solids: Per Cent. Wt.	Solids: Cum Per Cent. Wt.	Assay: Per Cent Total Cu	Assay: Per Cent Oxidized Cu	Per Cent. Wt. × Per Cent. Cu	Copper: Per Cent of Total	Copper: Cum Per Cent of Total
+ 20							
+ 28							
+ 35							
+ 48	0 5	0 5	2 22		1.11	0.04	0 04
+ 65	0 6	1 1	14 72		8 83	0 32	0 36
+100	1 6	2 7	15 82		25.31	0.90	1 26
+150	7.9	10 6	20 16		159 26	5.64	6 95
+200	10 6	21.2	23 99		254.29	9.08	16.03
−200	78 8	100 00	29 83		2,350.60	83 97	100 00
Total	100.00				2,799.40	100.00	
Check			27 84	1 44	Fe 19 8	Ins. 18 8	

Vanner Tails

Mesh	Solids: Per Cent Wt	Solids: Cum. Per Cent Wt	Assay: Per Cent Total Cu	Assay: Per Cent. Oxidized Cu	Per Cent. Wt × Per Cent. Cu	Copper: Per Cent. of Total	Copper: Cum Per Cent of Total
+ 20							
+ 28	1 4	1 4	0 66		0 92	2 11	2 11
+ 35	4 4	5 8	0 60		2 64	6 01	8 15
+ 48	9 9	15 7	0 60		5 94	13.61	21 76
+ 65	11 0	26 7	0 51		5 61	12 85	34 61
+100	16 6	43 3	0 48		7 97	18 25	52 86
+150	10 9	54 2	0.37		4 03	9.23	62.09
+200	6.3	60.5	0.37		2.33	5.34	67.43
−200	39 5	100 0	0 36		14 22	32 57	100 00
Total	100.0				43.66	100.00	
Check			0 41	0 25			

Vanner Concentrates

Mesh	Solids: Per Cent. Wt.	Solids: Cum Per Cent Wt	Assay: Per Cent. Total Cu	Assay: Per Cent. Oxidized Cu	Per Cent Wt × Per Cent Cu	Copper: Per Cent. of Total	Copper: Cum Per Cent. of Total
+ 20	0.2	0.2	2.02		0.40	0.04	0 04
+ 28	0 6	0 8	3 63		2.18	0.24	0.28
+ 35	1.6	2 4	7 26		11 64	1.28	1.56
+ 48	4 7	7 1	9.08		42 68	4.70	6.26
+ 65	8 2	15 3	10.90		89 38	9.84	16.10
+100	17.4	32 7	11.60		201 84	22 23	38 33
+150	17.6	50.3	11.60		204.16	22.49	60 82
+200	13.4	63.7	8.26		110 68	12 19	73.01
−200	36 3	100 0	6.75		245.02	26 99	100 0
Total	100.0				907.96	100.0	
Check			7.96	1 91			

Table Tails

Mesh	Solids: Per Cent Wt	Solids: Cum Cent Per Wt	Assay: Per Cent Total Cu	Assay: Per Cent Oxidized Cu	Per Cent Wt × Per Cent. Cu	Copper: Per Cent of Total	Copper: Cum Per Cent of Total
+ 20							
+ 28	0.9	0 9	0 65		0 59	1 26	1 26
+ 35	3.1	4 0	0 66		2 05	4 36	5 62
+ 48	7 2	11 7	0 60		4.32	9 20	14 82
+ 65	9.8	21.0	0 58		5.68	12.09	26 91
+100	14.8	35 8	0.54		7 99	17.01	43 92
+150	11.7	47 5	0 44		5 15	10 96	54 88
+200	7 1	54 6	0 49		3.48	7 41	62 29
−200	45 4	100 0	0 39		17 71	37 71	100 0
Total	100.0				46 97	100 0	
Check			0 41	0 24			

Table Concentrates

Mesh	Solids: Per Cent Wt.	Solids: Cum Cent Per Wt	Assay: Per Cent Total Cu.	Assay: Per Cent Oxidized Cu	Per Cent Wt × Per Cent. Cu	Copper: Per Cent of Total	Copper: Cum Per Cent of Total
+ 20	0 7	0.7	1.53		1.07	0.11	0.11
+ 28	1 0	1 7	4 23		4 23	0 45	0 56
+ 35	2 0	3 7	7 75		15 50	1 66	2 22
+ 48	6 3	10 0	9.79		61 68	6 59	8.81
+ 65	14.3	24.3	10 20		145 86	15 58	24.39
+100	36.8	61.1	8.77		322.74	34.48	58 87
+150	24.0	85.1	9.79		234.96	25.10	83.97
+200	8 9	94 0	10 81		96 21	10 28	94 25
−200	6.0	100.0	8.97		53.82	5 75	100.00
Total	100 0				936 07	100 0	
Check			8 96	1 40	Ins. 43 6		

Screen Analysis, Callow Machine

Mesh	Machine Tails							Machine Concentrates							Vanner Tails						
	Solids		Assay		Per Cent Wt × Per Cent. Cu	Copper		Solids		Assay		Per Cent. Wt. × Per Cent Cu	Copper		Solids		Assay		Per Cent. Wt. × Per Cent. Cu	Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent Oxidized Cu		Per Cent. of Total	Cum. Per Cent. of Total	Per Cent. Wt.	Cum Per Cent. Wt.	Per Cent. Total Cu.	Per Cent. Oxidized Cu		Per Cent. of Total	Cum Per Cent of Total	Per Cent. Wt.	Cum. Per Cent. Wt	Per Cent. Total Cu	Per Cent Oxidized Cu		Per Cent. Total	Cum. Per Cent of Total
+ 20	...	...	...	...	...	...	...	...	...	...	...	...	...	...							
+ 28	0.8	0.8	0.94	...	0.75	1.26	1.26	...	...	...	...	...	...	...	0.4	0.4	0.61		0.24	0.61	0.61
+ 35	1.2	2.0	0.81	...	0.97	1.63	2.89	...	...	...	...	...	...	...	1.8	2.2	0.53		0.95	2.40	3.01
+ 48	5.9	7.9	0.81	..	4.78	8.03	10.92	...	...	...	...	...	...	...	7.1	9.3	0.49		3.48	8.81	11.82
+ 65	10.4	18.3	0.87	...	9.05	15.21	26.13	0.6	0.6	5.14	...	3.08	0.48	0.48	11.4	20.7	0.49	...	5.59	14.15	25.97
+100	16.8	35.1	0.87	...	14.62	24.57	50.70	1.7	2.3	13.20	...	22.44	3.50	3.98	17.7	38.4	0.54	..	9.56	24.20	50.17
+150	12.5	47.6	0.80	...	10.00	16.80	67.50	8.8	11.1	19.15	...	168.52	26.25	30.23	12.1	50.5	0.44	...	5.32	13.47	63.64
+200	7.8	55.4	0.65	...	5.07	8.52	76.02	10.1	21.2	24.29	...	245.33	38.22	68.45	7.1	57.6	0.29	...	2.06	5.22	68.86
−200	44.6	100.0	0.32	...	14.27	23.98	100.0	78.8	100.0	25.70	...	2,025.16	31.55	100.00	42.4	100.0	0.29	..	12.30	31.14	100.0
Total...	100.00	.....	.....	.....	59.51	100.0	......	100.0	......	......	...	2,464.53	100.0	......	100.0	.....			39.50	100.0	......
Check	.	.	0.66	0.23	.	.	...	.....	.	24.80	1.24	Fe 19.4	Insol 24.6	.	.	.	0.37	0.15	...	. .	.....

Mesh	Vanner Concentrates							Table Tails							Table Concentrates						
	Solids		Assay		Per Cent. Wt. × Per Cent. Cu	Copper		Solids		Assay		Per Cent. Wt. × Per Cent. Cu	Copper		Solids		Assay		Per Cent Wt × Per Cent. Cu	Copper	
	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu		Per Cent. of Total	Cum Per Cent of Total	Per Cent Wt.	Cum Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cn		Per Cent of Total	Cum. Per Cent of Total	Per Cent. Wt.	Cum. Per Cent. Wt.	Per Cent. Total Cu	Per Cent. Oxidized Cu		Per Cent of Total	Cum Per Cent of Total
+ 20	0.4	0.4	2.42	....	0.97	0.15	0.15														
+ 28	0.6	1.0	1.81	...	1.09	0.17	0.32	0.5	0.5	0.88	...	0.44	1.10	1.10	0.8	0.8	3.67	...	2.94	0.34	0.34
+ 35	1.4	2.4	3.12	...	4.37	0.66	0.98	1.3	1.8	0.64	....	0.83	2.08	3.18	0.5	1.3	5.91	....	2.96	0.35	0.69
+ 48	3.3	5.7	6.35	....	20.96	3.18	4.16	4.2	6.0	0.57	....	2.39	5.98	9.16	5.2	6.5	9.59	..	49.87	5.82	6.51
+ 65	6.0	11.7	10.40	....	67.40	9.48	13.64	10.4	16.4	0.47	...	4.89	12.25	21.41	17.7	24.2	9.59	....	169.74	19.80	26.31
+100	16.0	27.7	12.81	....	204.96	31.13	44.77	16.0	32.4	0.44	....	7.04	17.63	39.04	40.0	64.2	8.06	....	322.40	37.61	63.92
+150	15.0	42.7	12.10	....	181.50	27.57	72.34	11.9	44.3	0.47	.	5.59	14.00	53.04	21.8	86.0	8.57	....	186.83	21.80	85.72
+200	12.1	54.8	6.76	....	81.80	12.42	84.76	7.8	52.1	0.44	..	3.43	8.59	61.63	8.2	94.2	8.87	. .	72.73	8.48	94.20
−200	45.2	100.0	2.22	....	100.34	15.24	100.0	47.9	100.0	0.32	....	15.32	38.37	100.0	5.8	100.0	8.57	....	49.71	5.80	100.0
Total...	100.0	......	......	.....	658.39	100.0	......	100.0	.....	.....	....	39.93	100.0	......	100.0	......	......	....	857.18	100.0	......
Check	....	.....	6.40	2.13	.....	.	. .	...	. .	0.45	0.25	. .	. .	..	...	..	7.84	1.44	Fe 21.0	.	. .

Table 2.—*Comparative Results Obtained in Operatian of C-B and Callow Flotation Machines at Concentrator No. 6, Arizona Copper Company, Ltd., Morenci, Ariz.*

May, 1915

		C-B Per Cent.	Callow, Per Cent.
Flotation machine tails	Total copper	0.72	0.81
	Oxidized copper	0 38	0 30
	Sulphide copper	0 34	0 51
Flotation machine concentrates	Total copper	38 19	24 82
	Insolubles	25.00	27.00
Vanner tails	Total copper	0 53	0 46
	Oxidized copper	*	0.18
	Sulphide copper	*	0.28
Vanner concentrates	Total copper	8.52	8.89
	Insolubles	* ...	30 60

June, 1915

		C-B Per Cent.	Callow, Per Cent.
Flotation machine tails	Total copper	0.71	0 69
	Oxidized copper	0 30	0 28
	Sulphide copper	0.41	0.41
Flotation machine concentrates	Total copper	35 24	25.41
	Insolubles	23.80	26 00
Vanner tails	Total copper	0 41	0 42
	Oxidized copper	0.23	0.22
	Sulphide copper	0.18	0.20
Vanner concentrates	Total copper	9 35	9.51
	Insolubles	*	* ....

July, 1915

		C-B Per Cent.	Callow, Per Cent.
Flotation machine tails	Total copper	0 61	0 66
	Oxidized copper	0.24	0.23
	Sulphide copper	0.37	0.43
Flotation machine concentrates	Total copper	27.84	24.80
	Insolubles	18.80	24.60
Vanner tails	Total copper	0.41	0.37
	Oxidized copper	0.25	0.15
	Sulphide copper	0.16	0.22
Vanner concentrates	Total copper	7.96	6.40
	Insolubles	*	* ....
Table tails	Total copper	0.41	0 45
	Oxidized copper	0.24	0.25
	Sulphide copper	0.17	0 20
Table concentrates	Total copper	8.96	7.84
	Insolubles	43.60	* ....

* No assay.

	C-B	Callow
Daily tonnage rate, average for the month of July, 1915	479	139

See Note, p. 666.

same simple reagents used elsewhere would apply. Further, a new type of pneumatic flotation machine, well adapted to the conditions at No. 6 Concentrator, had been successfully developed and its performance demonstrated on a full-sized unit. The new type machine would not be in any way embarrassed by the oversize coming from Hardinge mills.

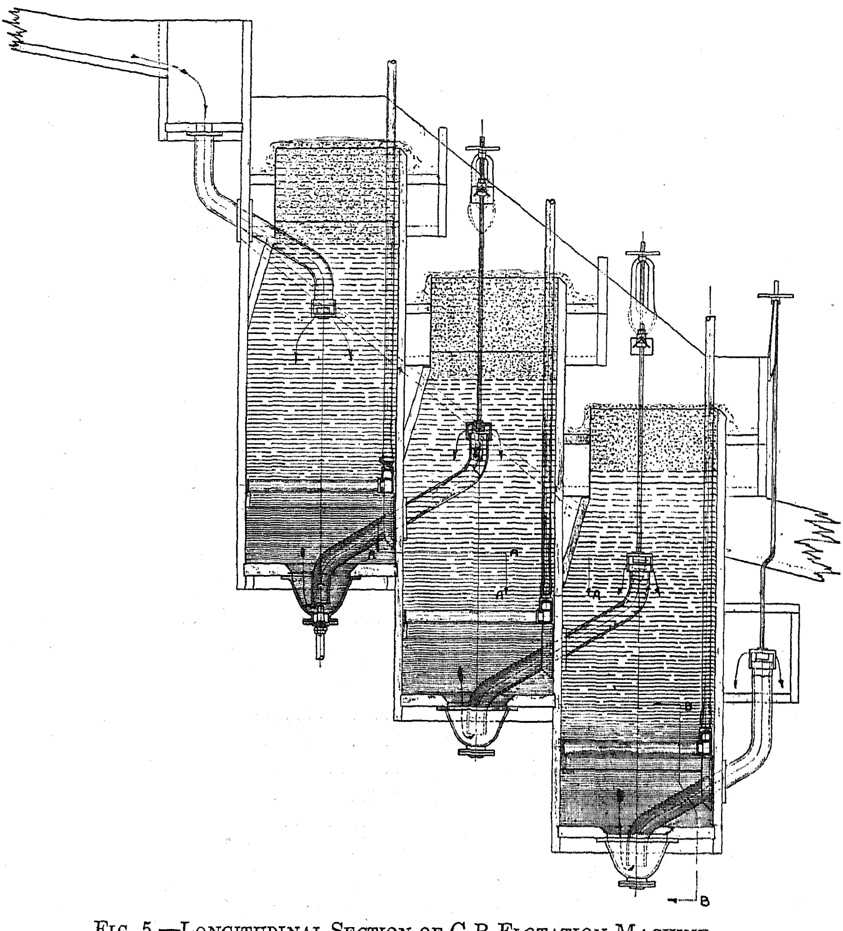

FIG. 5.—LONGITUDINAL SECTION OF C-B FLOTATION MACHINE. THREE CELLS IN SERIES.

All of the drag-belt overflow being handled by the colloid separator could go to a few of these new machines where all of the rich slimed sulphide copper would be taken out. Or the whole tonnage of reground

NOTE TO TABLE 2.—Without protection against "blinding" of the air-emitting media by sand or "oversize" the C-B machine handled considerably more than three times the tonnage handled by the Callow in July, and did this without detriment to the metallurgical work.

material produced in the Hardinge mills could go directly to the new type frother in which the slimed copper sulphides would be removed. The thoroughly frothed sands could then be treated on tables and vanners for the removal of the remaining sulphide particles which were too coarse to be separated by flotation. Since there would then be no embarassing losses in the slime part of the feed, these machines would work efficiently, and a maximum mill recovery would result.

After it became evident that flotation would apply to Morenci ores and before the value of the tube-grate idea was fully demonstrated, it was decided to install and experiment with a standard Callow flotation unit of 200 tons capacity, consisting of four rougher cells and one cleaner. This equipment was not received until after the full-sized C-B unit of 400 tons daily capacity had been installed.

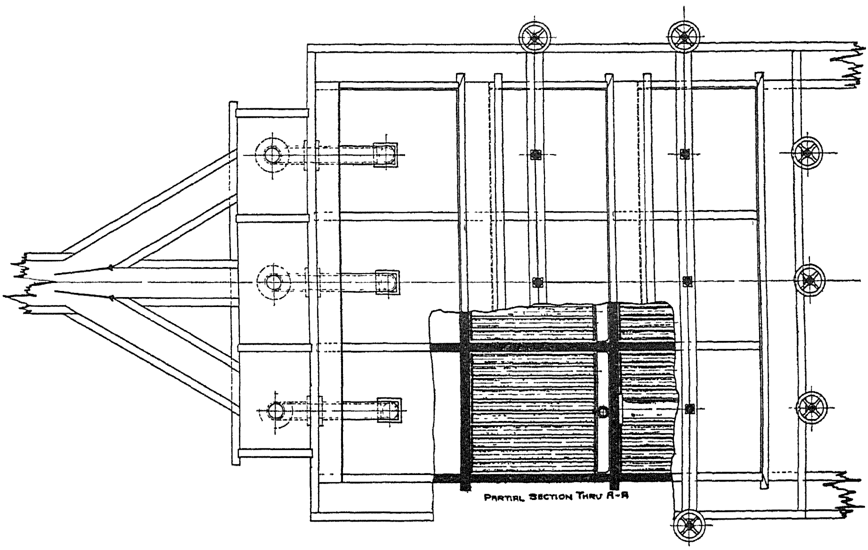

FIG. 6.—PLAN OF C-B FLOTATION MACHINE.

The Callow equipment was started in operation May 24, 1915, and competitive operation was carried on for about 3 months. The recoveries proved to be very much alike although the feed was not identical. The Callow apparatus is not adapted to handle coarse particles of feed or oversize, and had to be protected by a screen or spitzkasten. It will handle about one-half the normal tonnage of the C-B machine, occupying the same mill space. A summary of the results obtained for the months of May, June, and July, 1915, also details showing the work for the month of July, are given in Tables 1 and 2. The performance is shown to be substantially parallel as to quality of work done, but quite different as to quantity of tonnage handled.

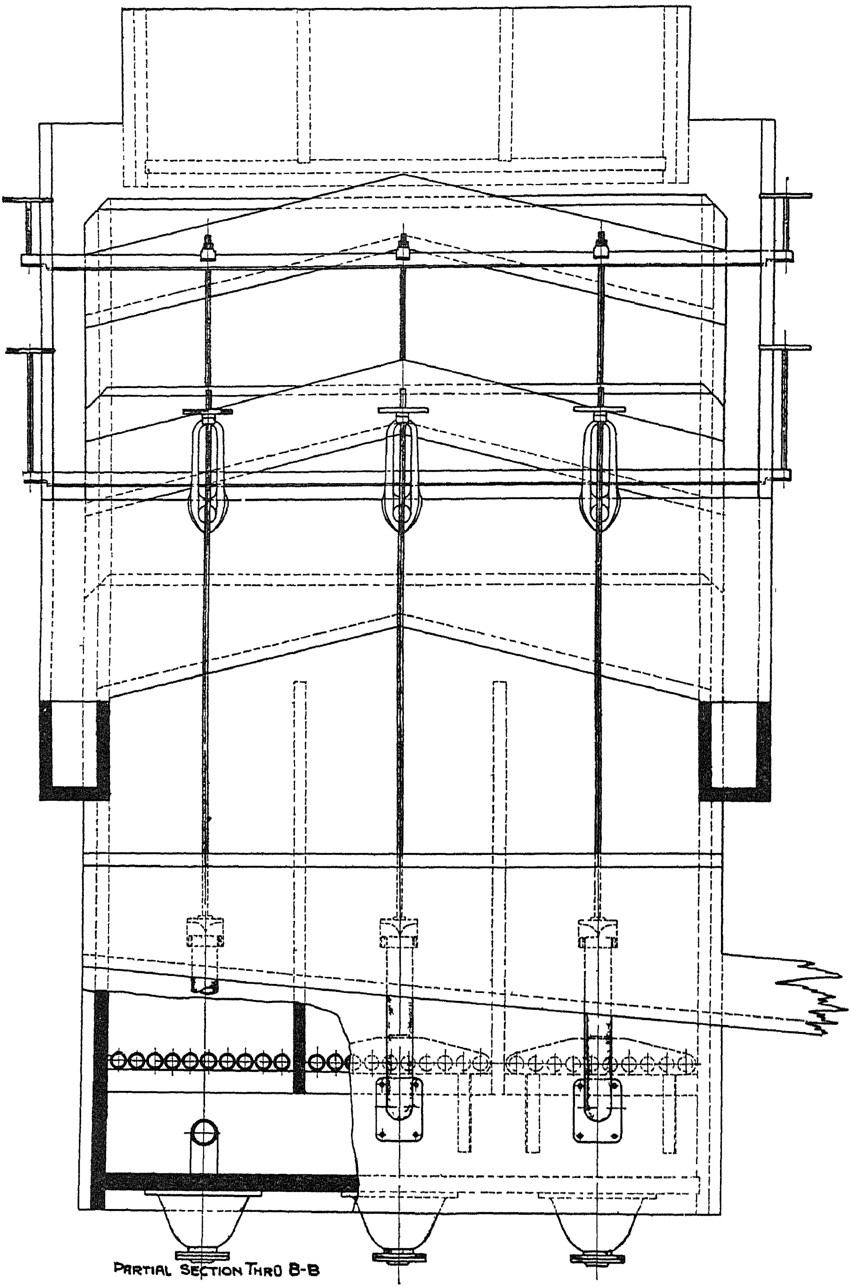

FIG. 7.—FRONT ELEVATION OF C-B FLOTATION MACHINE.

As mentioned before, the air and power consumption was not determined in the C-B installation because there was no way to take even approximately correct measurements. But the experience at Inspiration, where the C-B and Callow systems were also being operated in parallel, showed that substantially the same amounts of air and power were used by each system.

Experience suggested wider launders for froth, larger cleaner capacity, and simplified tube-grate construction for the C-B unit. These ideas are incorporated in the new design shown in Figs. 5, 6 and 7 in which it will be noted that the machine is merely a stationary wooden box suitably arranged to receive air-emitting tubes which are dropped in from the top

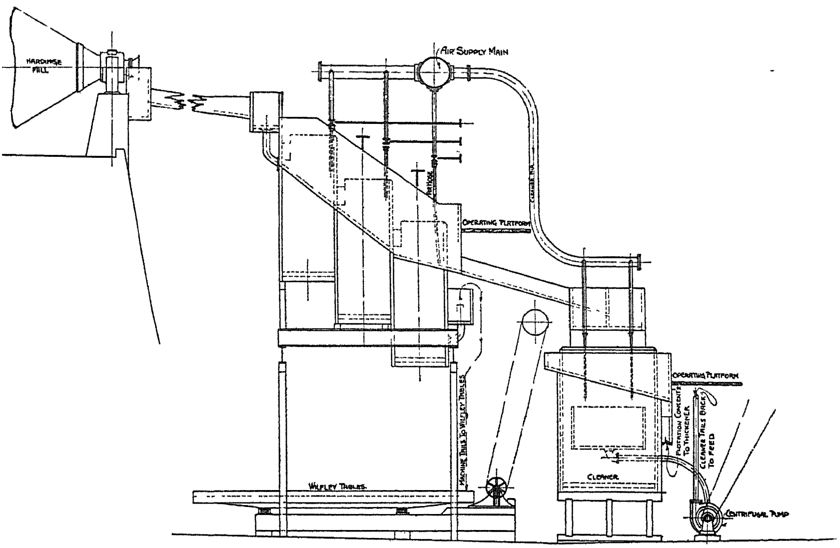

FIG. 8.—ARRANGEMENT AT MILL OF CANANEA CONSOLIDATED COPPER CO., CANANEA, MEX., USING THE C-B 3-STAGE FLOTATION MACHINE.

and rest upon ledges at the proper level in the pulp to be treated. These air-emitting elements are connected to the air supply by the use of rubber hose. They can be taken out or put in without cutting off the feed or shutting down the machine, in case this should be of advantage. An air pressure of 5 lb. is required. The air-emitting surface in the C-B machine is more than twice the complete cross-sectional area of the frothing compartments, and even if a ridge of sand should lodge upon the extreme top of the tubes and partially cut off the air supply, the remaining unobstructed area would still be larger than the whole cross-section.

Fig. 8 shows the improved form with cleaners as arranged in the mill of the Cananea Consolidated Copper Co., and is typical of the arrangement adopted for the later models. There are eight units in this installation. They are operated under conditions varying greatly as to quality

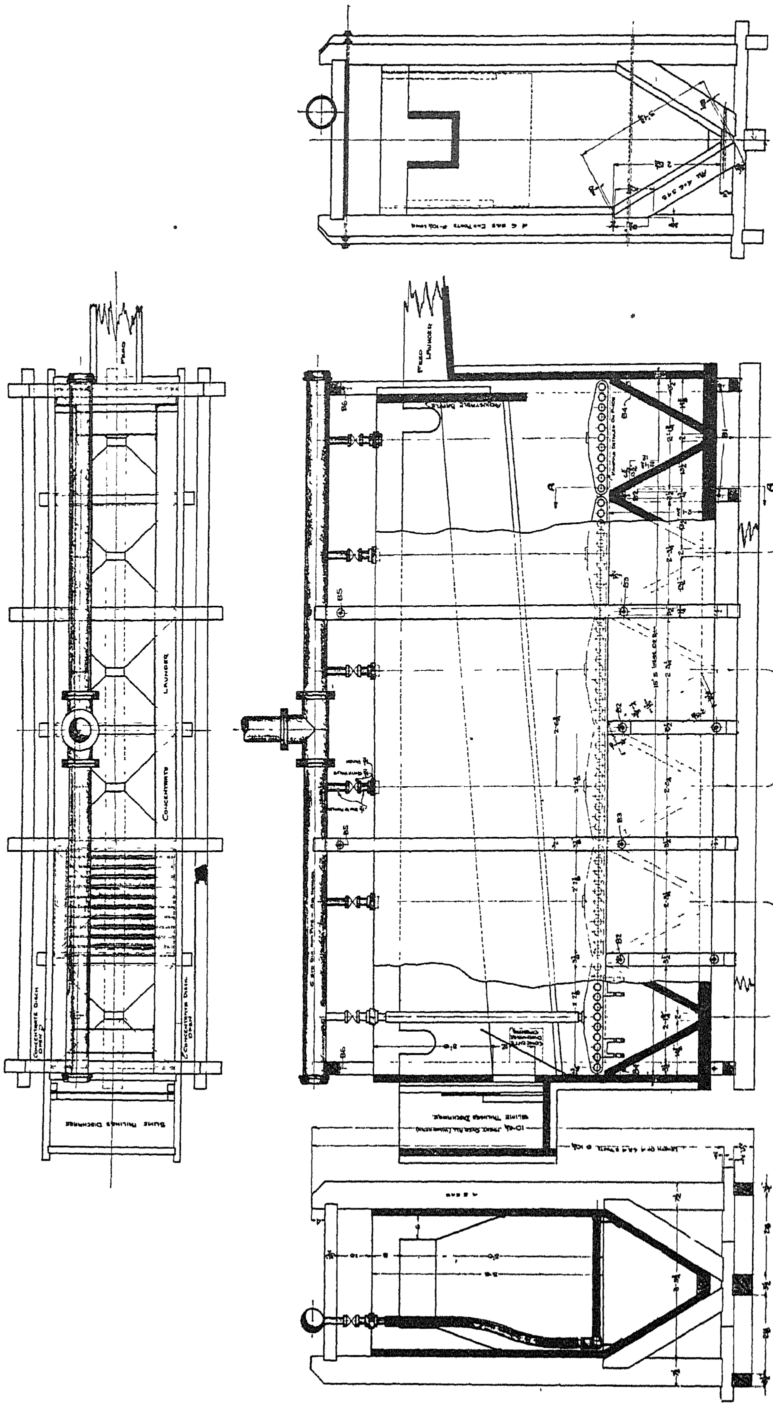

FIG 9.—SPITZKASTEN FROTHER EQUIPPED WITH TUBE-GRATE AIR FILTER.

of ores, there being a wide range of iron and copper sulphide conditions and all conditions of oxidation in the ore handled. The operation of the Cananea machines has been repeatedly interrupted by the internal strife in Mexico. Low "sulphide" copper contents in the tailing produced is common and the plant operates very satisfactorily, but no deductions of value for publication are at present available.

Fig. 9 shows an application of the tube-grate idea to a spitzkasten type of frother. One of these machines was made and installed in No. 6 Concentrator at Morenci but was taken out before it was tried. It seems to embody advantages of much promise in a frothing first flow sheet and will soon have a trial to determine its value in the simplified concentration of ore that is amenable to flotation.

A Combined Hydraulic and Mechanical Classifier

BY M. G. F. SÖHNLEIN,* MACHACAMARCA, BOLIVIA, S. A.

(Arizona Meeting, September, 1916)

In a Bolivian tin concentrator an appliance was needed to furnish a suitable product for fine jigging from a pulp of the following composition:

Mesh	Per Cent.
+ 20	8.0
+ 40	36.5
+ 60	9.0
+ 80	10.5
+100	5.5
+150	4.5
−150	25.0
	99.0
Loss	1.0

In the jigging process, particles of cassiterite as small as 0.1 mm. can be recovered, provided they are not present in excess, that is if the material treated on the jigs contains sufficient coarser sands to keep the interstices between the grains open. Removing the fine material from the pulp by screening, to prepare the jig feed, is impractical, because that means the use of an 80-mesh screen or finer, and is not as effective as hydraulic classification. A screen would eliminate mineral grains from the feed which can be recovered by jigging, and in addition would throw more gangue and low-grade middling on to the jig.

Formerly a one-spigot Richards vortex classifier had been used to accomplish this separation, but the tangential openings through which the hydraulic water enters became clogged by fine vegetal matter with which the water was contaminated, and which it was impossible to remove from the water. Consequently the work of the classifier was imperfect and a good deal of slime was sent to the jig. When designing the plant referred to in this paper it was decided to use a hydraulic classifier as shown in Fig. 1. The essential features had been copied from the Anaconda classifier.[1] This remodeled classifier consists of a truncated pyramidal wooden hopper *A* with an attachment of cast iron

* Mill Supt., Compañía Minera de Oruro.

[1] *Trans.*, vol. 46, p. 277 (1913).

bolted to it at the bottom. The heavier and larger particles settle and pass through the 1¼-in. nipple *C* which is screwed into a flange and laid loose in a recess turned in the casting *B*. The hydraulic water enters through the 1½-in. pipe *D* which is screwed into the casting. The distance of the pipe from the exit of the nipple *C* can be regulated by screwing the pipe up or down, a short hose connection being provided between the valve *E* and the mill water piping to make this flexible.

The work of this classifier was satisfactory if sufficient hydraulic water was used, but under these conditions a good deal of material that should have gone to the jig was carried into the overflow. The difficulty could have been remedied by again submitting the overflow to hydraulic classification in a second pocket, but this was objectionable because it

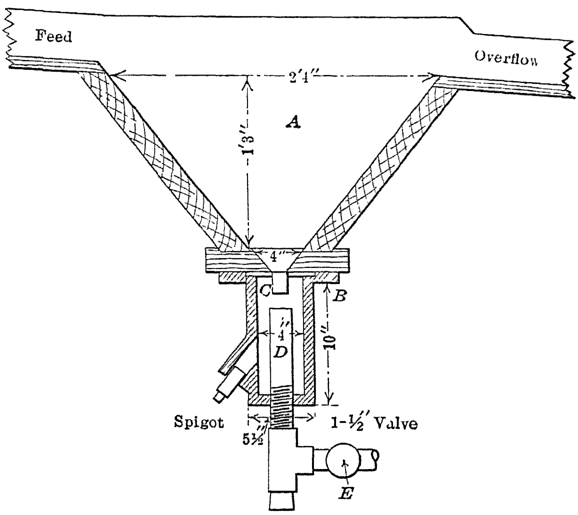

Fig. 1.—Sectional Elevation of Hydraulic Classifier.

would have added another quantity of hydraulic water to the pulp, and would have diluted the overflow too much. The classifier was therefore run with less hydraulic water, so that its sand product contained some slime. The separation effected by the jig was not quite satisfactory under these conditions, since the large amount of fines in the feed made the beds pack in hard banks, producing a low-grade concentrate and causing free mineral to go into the tailing.

The desideratum in this instance was to obtain separation of the part of the sands in the pulp under hindered-settling conditions, or nearly so, without diluting the fine material with too much water. The sand product from a mechanical classifier, although practically free from slime, is the result of free settling of the pulp, and therefore is not as desirable as the sand separated by hindered settling in a hydraulic device. I

therefore designed an apparatus which combines the advantages of the hydraulic and the mechanical classifier. Having previously used a paddle-wheel of the Fleming type for dewatering, this machine was selected as a mechanical classifier in preference to the more generally used types on account of its compactness and its low first cost.

As shown in Fig. 2, the device consists of a narrow wooden trough in which the paddle-wheel revolves. Details of the construction of the

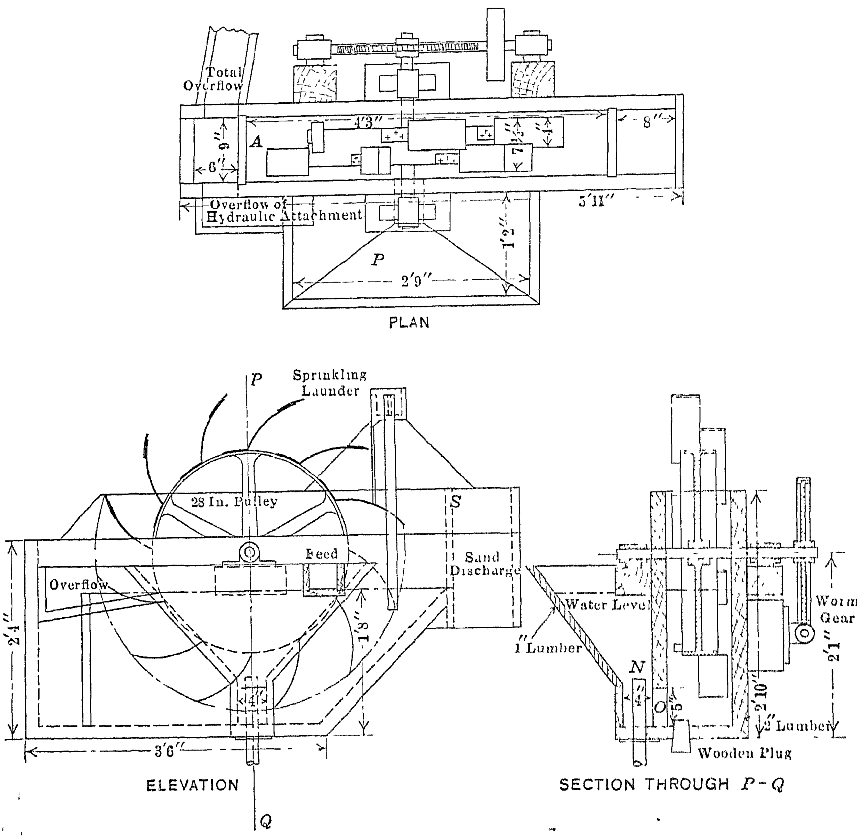

FIG. 2.—COMBINED HYDRAULIC AND MECHANICAL CLASSIFIER.

classifier can be easily gathered from the accompanying drawing. The tips of the blades are protected by ¼-in. iron strips fixed with two countersunk rivets. The strips last about four months and are replaced at small cost. The blades are cut from ¼-in. plate and bent in the fire. They are fixed to the rim of the pulley by three 1 by ⅜-in. machine bolts. The holes in the rim and those in the blades are bored from a template and spare blades are kept ready so that little time is lost in making renewals. The wheel is driven at the rate of 3 r.p.m. by a simple worm

gear—in this case taken from an old vanner—which is directly belted to a line shaft. The feed is introduced into the trough near the bottom at O, after having been submitted to hydraulic classification in the pyramidal attachment P in which an ascending current of water flows out of the pipe N. Consequently the material which is undesirable in the jig feed is floated off with the use of less hydraulic water than needed in an ordinary hindered-settling classifier because the slime that comes with the sand into the trough is separated by the paddle-wheel and flows over at A. It is evident that classification in the hydraulic pocket does not entirely take place under hindered-settling conditions, because in that case the sand treated by the wheel could not contain any slime. The amount of hydraulic water used is such that the dilution of the overflow does not exceed the required density and therefore the classification cannot be perfect. However, the most objectionable factor, namely, the presence of slime in the jig feed, is to a large degree eliminated by the subsequent action of the paddle-wheel. The moisture in the sand discharged by the wheel can be regulated in two different ways, (1) by raising the level of discharge, and (2) by increasing the distance between the tips of the blades and the discharge.

The discharge level of the sand product can be varied by inserting strips of wood of different heights into the slots marked S. To obtain practical results the sand is discharged at a point 1½ in. above the overflow with 50 to 55 per cent. moisture, but with the level of sand discharge raised to 2½ in. above the overflow, and the discharge edge spaced from the wheel as shown in the drawing, the product contains but 30 to 35 per cent. moisture, but under these conditions the capacity decreases out of all proportion. If the machine is overfed, the sand piles up so high at the discharge that part of it remains lying on the paddles, and falls back into the trough at the other side, finally choking the classifier.

The second method gives a drier and cleaner product, because the sand is piled up in front of the wheel until it reaches sufficient height to slide over the discharge edge, and nearly all the water drains back into the trough. There is no objection to placing the overflow rim in the trough close to the paddles, which makes the machine more compact. I formerly tried to obtain a more thorough settling by leaving a distance of about 10 in. between the tips of the paddles and the overflow, with the only result that the space between became tightly packed with fine sand, and the actual overflow was formed at only ½ in. from the tips.

A spray is provided to remove the adhering grains of sand from the blades of the wheel, and after having cleaned the blades, the water falls on the pile of sand lying in front of the wheel and gives it a final wash. The slower the wheel revolves the cleaner will be the sand product, because then every blade brings up slightly more sand than it can deliver, and part of the material falls back into the trough to be shoved up again by the next

blade. Therefore the sand is repeatedly turned over before being discharged, which materially assists in removing the slime.

The classifier can easily handle 60 tons pér 24 hr. of dry pulp with a specific gravity of 3.5 to 3.8. Screen tests made on the products are shown below:

Mesh	Sand Product Per Cent.	Overflow, Per Cent.
+ 20	11.0	0 3
+ 40	42.0	5.4
+ 60	13 5	12 5
+ 80	16 3	11.0
+100	8 0	9 3
+150	2 5	11 5
−150 sand	4 2	48 2
slime	1 4	
	98 9	98 2
Loss	1.1	1 8

These figures show that there is no separation at a definite screen size, but this can not be expected when classifying a pulp that contains particles of cassiterite with a specific gravity of over 6.5 and particles of gangue with a specific gravity less than 3.0. The plus 60-mesh material in the overflow is all very low-grade, and the minus 80-mesh in the sand product is the richest material in this product.

The jigging practice has considerably improved with the better adapted feed and the grade of the concentrate has been raised from 66 per cent. metallic tin to an average of 70 per cent. and more. The concentrate from the jig, which is of the Harz type making a concentrate on three sieves, has the following composition:

Mesh	Per Cent.	Metallic Tin, Per Cent
+ 20	20.9	71.7
+ 40	55.3	72.2
+ 60	10.4	71.0
+ 80	4 6	65 6
+100	2 7	66 2
+150	3 6	70 1
−150	1 3	68.3
Total and average	98.8	71.36
Loss	1 2	
	100.0	

The feed to the classifier contained in this instance 3.5 per cent. tin, the sand product or feed to the jig was not assayed, but carried probably somewhat more cassiterite than the original pulp.

When working on a richer feed of about 13 per cent. tin, the concen-

trate from the jig contains a much larger percentage of fines, as is shown below:

Mesh	Per Cent.	Metallic Tin, Per Cent.
+20	10.0	71.8
+40	36.5	72.1
+60	16.5	72.7
+80	12.0	72 0
+100	6.5	69.8
+150	7.0	69.2
−150	10.0	70 3
Total and average	98.5	71.98

I will now explain why in this case jigging is used for the treatment of a pulp that could be handled by tables. When dressing an ore assaying less than 5 per cent. tin on tables it is impossible, even with the best classification, to obtain at one operation a concentrate assaying better than about 63 per cent. tin, whereas a jig gives a product of a considerably higher grade, and once it has been properly adjusted does not need much attention, even with a varying amount of feed. When working on richer material, a table does not produce a concentrate of more than 66 per cent. In tin dressing the grade of concentrate produced is of paramount importance, on account of heavy freight charges to the smelters and the reduced smelting charges on high-grade products. It is therefore advisable in every case to recover as much concentrate as possible on jigs.

The whole construction of the classifier could, of course, be improved, especially as regards the form and overflow of the hydraulic attachment, but since the machine does the work expected of it, it is not worth while to make changes. The power required to drive the wheel is under 1/6 hp. The only attention required is to lubricate the bearings occasionally and to adjust the water valves, if the quality of the feed varies. The machine has been working since the beginning of July, 1915, and at the date of writing (November, 1915) the tips of the paddles have just had to be renewed. One paddle has broken near the rim of the pulley, but this was probably due to poor material or perhaps because the paddle after bending had been cooled too quickly.

After writing this I noticed the classifier proposed by A. E. Drucker in the *Mining and Scientific Press* of Oct. 16, 1915, which is essentially the same as the paddle-wheel with hydraulic attachment. There is no reason why the latter machine could not be used with advantage to prepare the feed for a regrinding mill.

A Comparative Test of The Marathon, Chilean and Hardinge Mills

BY F. C. BLICKENSDERFER,* MORENCI, ARIZ.

(Arizona Meeting, September, 1916)

Introduction

During 1914 and 1915, extensive experiments were conducted at the concentrator of The Detroit Copper Mining Co. of Arizona, at Morenci, Ariz., in order to test the relative grinding efficiencies of the Marathon, the Chilean and the Hardinge Mills.

The Marathon mill used in the experiments was the first one ever built by the manufacturers; and since this was the first time that it was given a thorough tryout, with such remarkable results, the data have been assembled in the following paper and submitted in the interests of the milling profession.

Description of the Flow Sheet

The concentrator flow sheet, as it existed at the time of these tests, is shown in Fig. 1. The total mill feed, as delivered from the crushing plant, had as its maximum size 1-in. material. This material was screened dry on Cole Zig-Zag screens fitted with 4-mesh openings; the oversize going to one pair of coarse rolls, the undersize to five Wilfley tables fitted with Butchart National riffles. The coarse-roll product was elevated and screened wet on Zig-Zag screens fitted with 4-mesh openings, the undersize going to the five tables mentioned above and the oversize to one pair of fine rolls.

The fine-roll product was elevated and screened together with the coarse-roll product; so that, before any concentrating operations took place, the total mill feed was reduced to pass a 4-mesh screen.

After concentration on Butchart-riffled tables, the tailings from these tables passed to de-sliming hoppers, where two products were formed, *viz.*, an overflow and a discharge. The discharge constituted the feed for the Chilean mills. It contained 16 per cent. slime (−200 mesh) and 3 parts of water to 1 of solids. The concentrator was equipped regularly with four Chilean mills (Monadnock type) to the unit; but, in order to

* Concentrator Metallurgist, Detroit Copper Mining Co.

provide similar conditions for a test, two of these mills were replaced by a Hardinge mill and a Marathon mill. The Hardinge mill was fed with the discharge from the de-sliming hopper; but a greater proportion of the slime was removed in order to get a drier feed, containing but 8 per cent. slime (−200 mesh) and with a ratio of 1.3 parts water to 1 of solids. The Marathon mill was also fed with the discharge from the de-sliming

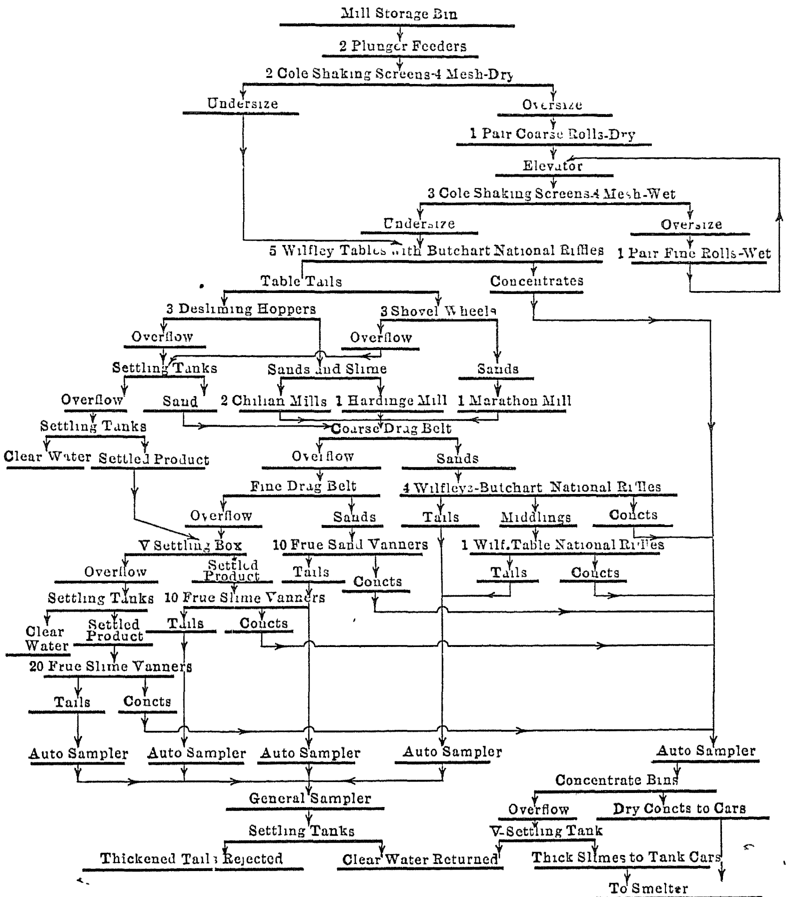

FIG. 1.—FLOW SHEET OF 750-TON CONCENTRATOR UNIT OF DETROIT COPPER MINING CO., MORENCI, ARIZ.

hopper, containing 12 per cent. slime and having a water to solids ratio of 1.8 : 1. Three shovel-wheels were temporarily installed to remove all of the slime from the table tails, for the purpose of giving a much larger tonnage to the Marathon mill, it being found that the total normal unit tonnage could be handled by the three shovel-wheels.

All of the mill products passed to a coarse drag belt which took out the coarse sand for treatment on the Butchart-riffled tables; the coarse

drag-belt overflow flowed to a fine drag belt, which removed fine sand for treatment on Frue sand-vanners. The overflow from the fine drag belt passed to a V-box which removed a thick product for treatment on Frue slime-vanners; the V-box overflow was thickened for final treatment on Frue slime-vanners. All rejected tailings were sampled separately as they came from each group of machines, then a general sample of the tails was taken after all of the tailings came together at a common point. The clear water at each point, if at a sufficiently high level, was used for wash water on machines below; but, if not, it was collected at the bottom of the concentrator and pumped to tanks at the top to be used over again.

The crude milling ore is a quartz-monzonite porphyry with chalcocite as the copper mineral. An average assay for one year is: Cu, 2.80; SiO_2, 59.5; Al_2O_3, 16.8; Fe, 5.0; S, 5.0; CaO, 0.7; and MgO, 0.9 per cent. The ore contains a small percentage of copper sulphate, which is soluble in water. This "acid water" is very destructive to all iron parts, so that iron consumed in all these tests is excessive to a small degree for quantities of feed crushed. Unslacked lime is added continuously at the rate of 2,000 lb. daily to circulating mill water to neutralize the copper sulphate.

General Conditions Regarding the Tests

These experiments were carried on for several months, so that results obtained are not spasmodic, but represent the average results of routine work of each mill under all the variable conditions encountered in actual milling practice.

The Chilean and Hardinge mills ran together for a continuous period of 63 days, or for the length of time required to wear out the tires and die of the Chilean mill. The Hardinge mill ran the first 30 days with an old lining. The old lining was then removed, a new one put in and the test completed.

The average results of the Hardinge and Chilean mills for the 63-day period will be used in comparison with the Marathon mill. Test No. 1 of the Marathon mill was run under the same conditions regarding feed as the Hardinge and Chilean, so that an excellent comparison is possible.

Some time elapsed between tests Nos. 1 and 2 of the Marathon mill, during which some preliminary tests were made to determine the best conditions for operating with a heavy tonnage. During this time the three shovel-wheels were installed to remove all slimes and the greater amount of water. Also about this time the tonnage per unit was reduced from 750 tons to 500 tons per 24 hr., and this total unit feed was sent through the Marathon mill. Subtracting the weight of concentrates produced in primary table concentration, and the weight of slimes removed by the shovel-wheels preceding the Marathon mill, a net dry

weight of 430 tons per 24 hr. was obtained. The total recovery of the unit under these conditions was almost exactly the same as the second unit operating three Chilean mills in the same step. This fact paved the way for a heavy tonnage on the Marathon mill, known in this report as test No. 2.

Description of the Three Types of Mills Tested

The Chilean mill used in the test was of the Monadnock type, 5 ft. in diameter, and fitted with 1½-mm. round-hole punched screens. The Chilean die (or ring), when new, weighed 1,722 lb., the three tires together weighed 2,526 lb. A 50-hp. motor furnished power to drive the Chilean mill, transmission taking place in two stages, viz., from motor pulley to line shaft, thence to Chilean mill belt wheel.

The cylindrical portion of the Hardinge mill was 8 ft. in diameter by 3 ft. long. It was lined with Danish pebbles of uniform size, set in concrete. The method of lining was as follows: The lower half was lined

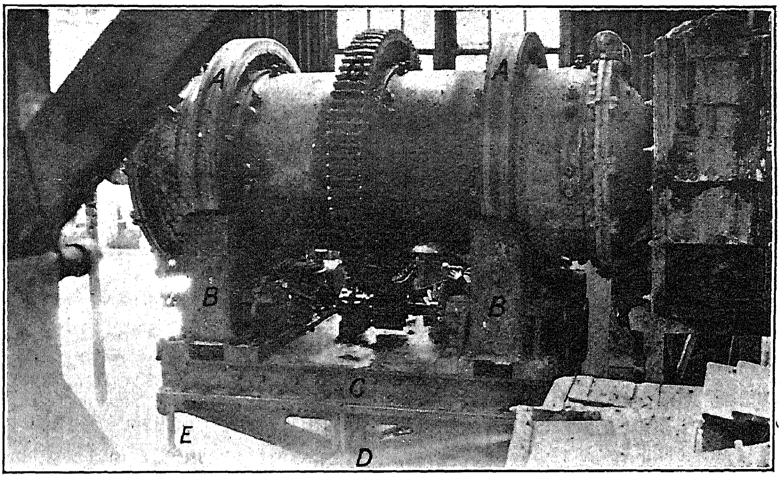

Fig. 2.—Old Marathon Mill.

first and allowed to "set," then the mill was turned over so that the unlined portion was below. This section was then lined and the total lining allowed to set 48 hr. The Hardinge mill was charged with 5 tons of Danish pebbles of fairly uniform size. Power was furnished by a 75-hp. motor direct-connected to a gear shaft which in turn engaged with the large gear wheel of the same diameter as the mill itself.

The Marathon mill consisted essentially of a tube, 3 ft. in diameter by 7 ft. long, surrounded by two tires, A (Fig. 2), near each end, these tires

being supported by and revolving upon two pairs of rollers, *B*. Two idler rollers, *H*, keep the tires in alignment with the rollers when the mill is in action. The four rollers *B* are set on frame *C* and are rigidly bolted to it. Frame *C* is pivoted at its center, *D*, thus providing a tilting arrangement, which permits of adjustment to any desired slope by bolts, *E*, at either end of the frame. The ends are bolted to the mill tube, and can be quickly and easily removed for the inspection or renewal of crushing parts. The feeding arrangement consists of an L-shaped tube, the horizontal leg of which discharges into the mill through a circular opening in the center of the head end. The product, in wet crushing, is discharged through a 6-in. pipe centrally located in the tail end. In dry crushing, the Marathon mill has a special end with slots in the periphery for the free discharge of the crushed product. A large hopper is set below the discharge end to collect the product.

The Marathon mill lining consists of 16 cast-iron plates bolted to the mill shell, the total weight of the plates being 4,480 lb. The crushing medium is composed of iron rods ranging in size from ½ in. to 2 in. in diameter; the initial charge of rods weighed 7,000 lb. The mill is driven by a gear wheel, *G*, connected by two sets of gears to a drive wheel, which is belted to a 25-hp. motor.

Modifications of Original Design of Marathon Mill

During the progress of the experimental work with the Marathon mill, several construction defects developed, which had to be remedied before further operation could proceed. A preliminary run with no load of ore in the mill, showed that the understructure was not rigid enough to prevent excessive vibration. The top and bottom of the frame were entirely covered with ½-in. and 5/16-in. sheet-steel plates respectively, the plates being firmly riveted to the legs of channel irons composing the top of the tilting frame.

The feed spout as furnished with the mill gave endless trouble with a wet feed, because the pulp was dashed out around the feed spout where it enters the center of the mill. The feed spout was replaced by a scoop feeder which was bolted to the head end of the mill.

A circular cast-iron liner (1¼ in. thick at the edges by 2¼ in. at the center) of the same diameter as the inside of the mill, was bolted to the head end to protect it against excessive wear.

The roller bearings supporting the axles of rollers were very unsatisfactory. Being of a spiral type, they were forced to extend by the weight of the mill to such a degree that the bearing end was thrust off. This extension of spiral rollers caused a reduction in their diameter, which resulted in a loose wabbly bearing. In case of such a failure the roller-bearing parts were renewed.

Methods of Operating the Three Types of Mills

The best method of operating the Chilean mill is perhaps too well known to require a detailed description. It is essential to have sufficient water in the feed for the Chilean mill to obtain best results from the mill screens, for if the pulp is too thick the screens clog and cause the mill to become overloaded. If too great a quantity of pulp is fed to the mill, overloading follows at once and the mill does practically no grinding. When the Chilean mill became overloaded, the power required to drive it was about one-third more than under normal conditions. An average ratio of 3 parts water to 1 part solids (25 per cent. solids) was the best pulp to feed to the Chilean mill for all-around good work. The speed was 40 r.p m.

The Hardinge required a less dilute pulp (42.8 per cent. solids) for best results. It is practically impossible to overload the Hardinge mill, "so to speak," as long as the mill is able to discharge the product, but constant attention is required to prevent "oversize." When its grinding capacity is exceeded, a considerable amount of very coarse material traverses the length of the mill without receiving a single blow from the falling pebbles. Special attention was given each of the mills in order to keep them up to full capacity with the production of a minimum quantity of "oversize." The mill was operated at 29 r.p.m.

The Marathon mill in Test No. 1 was operated as a fine grinding machine so as to give a product of the same fineness as the Hardinge and the Chilean mills. The most favorable results were obtained with pulp that contained 36.7 per cent. solids. When the Marathon mill became overloaded it was surprising to find no coarse oversize in the product, *but a product that was uniformly coarser.* This is a fact that requires special notice. It is equivalent to saying that if the mill is once started up it is not necessary to pay any attention to the product, for if overloaded, the whole product will be slightly coarser, and if underloaded the product will be slightly finer.

In test No. 2 of the Marathon mill the amount of water and slime in the feed was reduced to a minimum by shovel-wheels to secure a less volume of pulp and a consequent slower velocity of pulp through the mill. The tonnage of test No. 2 was greatly increased over that of test No. 1, and the Hardinge and Chilean mill tonnages. The feed contained an average of 63.5 per cent. solids, and only 2.92 per cent. slime (−200 mesh). The speed in all the tests of the Marathon mill was 30 r.p.m. In a diameter of 3 ft. at 30 r.p.m. there is no dead zone in the periphery of the mill, while excessive slipping of rods is prevented by having the liner plates thicker at each edge than in the middle, resulting in a corrugated interior surface. The lining retains this corrugated surface until it is worn out. The rods wear evenly from end

to end, do not get crosswise in the mill and are reduced to a diameter of ⅜ in. before they begin to crush, flatten out or break into pieces. When worn to this thinness, some rods roll up and are discharged; the short half-length pieces retain their positions in the charge of rods without any serious results, but may reduce crushing efficiency to some extent. It is good practice to take out the entire charge of rods every 10 days, sort out the small and "disabled" rods and replace the loss in weight with 1¼-in. rods. As soon as the weight of rods consumed per day was determined, their loss in weight was replaced daily with fresh rods. The mill must be closed down to add the daily charge of rods. During these tests the Marathon mill was tilted ¼ in. per foot, but later tests proved that it would produce equally as good results when run horizontally.

Some experiments were made on coarse (+ ½-in.) material both wet and dry, using the peripheral discharge end and feed spout. The charge of rods in the mill was increased by the addition of steel shafting 4 in. in diameter. These tests were not extended over any length of time, nor were sufficiently conclusive results obtained to warrant their mention. The primary fact established was this: The mill was able to handle coarse feed up to 2 in. in a way to encourage further experiment.

Explanation of the Action of the Marathon Mill

It might be well at this point to offer an explanation of the action that takes place within the Marathon mill.

For the sake of comparison, if we select any type of mill with spherical-shaped grinding media, the Hardinge mill for example, and consider a single crushing element, we have a row of pebbles in contact with another row of pebbles as indicated in Fig. 3.

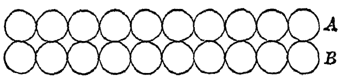

FIG. 3.

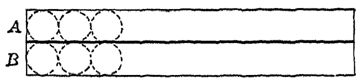

FIG. 4.

Now when falling, row *A* strikes row *B*, or if rolling together, row *A* rolls on row *B*; it is evident that crushing takes place where one pebble strikes or rolls against another, or at their point of contact. The number of points of contact in an element is determined by the size of pebbles; the smaller the size of pebbles, the greater the number of points of contact and *vice versa*. But it has been demonstrated in actual practice that the pebbles must be above a certain size to develop, when falling, a crushing blow of sufficient force to be of any value. Hence crushing ability depends not only on points of contact, but also on the size of pebbles and on their mass (by *mass* is meant specific gravity).

Since the number of points of contact must vary inversely as the size of pebbles, the two factors are compromised by using various sizes of pebbles. In order to take advantage of the mass factor, iron balls have been recently substituted for pebbles as a crushing medium.

Now consider a solid bar for the grinding medium, in the case of the Marathon mill, as shown in Fig. 4. When falling, rod *A* strikes rod *B*, or if rolling together, rod *A* rolls on rod *B;* there is a continuous row of points of contact, or geometrically speaking, a line. It has been shown in experiments that the rods remain parallel when the mill is in operation. Hence we have the maximum number of contact points that

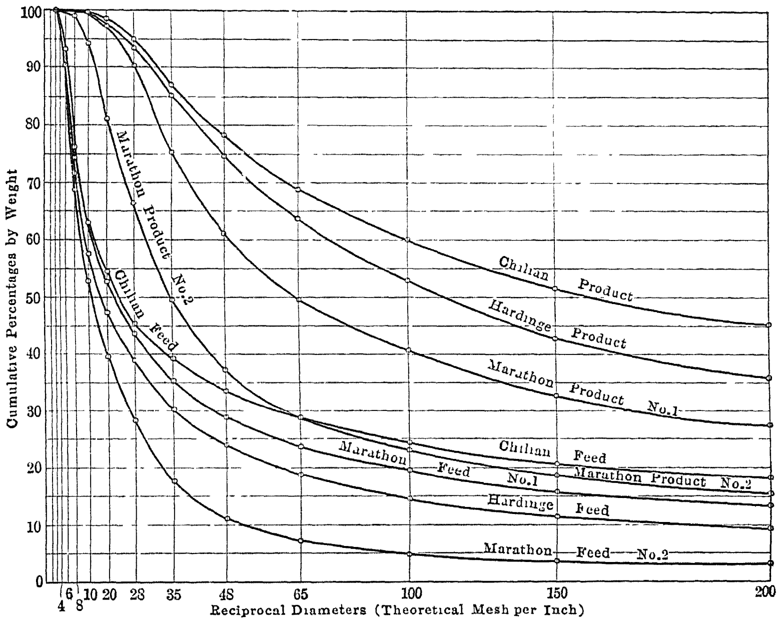

FIG. 5.—DIAGRAM OF CRUSHING EFFICIENCY—GATES METHOD.

can exist between the surfaces of two rods (cylindrically shaped). Again, we may consider the two rods *A* and *B* of Fig. 4 as made up of solid iron balls packed closely and bound together throughout their length by additional iron which is 0.4764 of their own weight. Therefore, the crushing ability of iron rods—bearing in mind that crushing ability depends upon points of contact and upon the mass of crushing medium—must be greatly superior to those mills which use spherical-shaped crushing media. The relatively small diameter of rods, 2 in. to ⅜ in., causes them to fit compactly, thus requiring a minimum falling distance to deliver a crushing blow. The center of mass of a charge of rods is much nearer the axis of the mill than in a mill of greater diameter whose crushing charge consists of steel balls or pebbles.

It is necessary in ball and pebble mills to have a sufficient weight of charge to deliver a crushing blow. This condition is effected either by making the mill of large diameter or by increasing its length. When the length is increased the power factor increases, when the diameter is increased the power factor increases very rapidly and at the same time there is danger of the peripheral speed becoming too great, tending to form a dead zone.

Tables of Results Showing Items and Conclusions

Table 1.—*Cost Comparisons*

	Marathon Test No 1, Per Cent.	Marathon Test No 2, Per Cent	Hardinge Mill, Per Cent	Chilean Mill, Per Cent.
Operating Labor	13 38	11 21	7.75	11 63
Labor on Repairs	12.20	12.27	3.85	7.43
Material on Repairs	40.38	42.29	28 40	22 52
Power	34 04	34.23	60 00	58.42
	100 00	100 00	100 00	100 00

Marathon No. 1 is Exceeded in Operating Costs by	Excess Cost, Per Cent
Marathon No. 2	19.34
Hardinge	72 56
Chilean	15.00

Chilean is Exceeded in Operating Costs by	Excess Cost, Per Cent.
Marathon No 2	3.77
Hardinge	50.04

Marathon No 2 is Exceeded in Operating Cost by	Excess Cost, Per Cent.
Hardinge	44.59

Costs are generally governed by local conditions. In each of the divisions, Operating Labor, Labor on Repairs, Material on Repairs, and Power, the percentage of the total cost of the mill is represented, no correction being made for the tonnage. From the second part of this table it is to be noted that the Marathon mill costs as operated in test No. 1 are the least; the Chilean mill costs are the next lowest; then the Marathon mill costs as operated in test No. 2; and the Hardinge mill costs are the highest. In comparing the costs of one mill with those of any other, for instance the Hardinge costs and the Chilean costs, the mill with the lowest costs is taken as 100 per cent. The Hardinge costs on this basis are 150.04 per cent., or 50.04 per cent. in excess of the Chilean costs.

Material on Repairs includes rods and liners for the Marathon mill, die and tires for the Chilean mill, and pebble lining and pebbles for the Hardinge mill.

In each case the power factor is strongly in favor of the Marathon mill. The tonnage factor is omitted at this point because it properly belongs to a comparison of grinding efficiencies.

TABLE 2.—*Record of the Average for Each Test*

Type of Mill	Running Time in Per Cent. of Total Time	Time Lost, Per Cent.	Total Actual Running Time in Hours	Tons Dry Feed in 24 Hr.	Tons Dry Feed per Hour	Tons Pulp in 24 Hr.	Tons Pulp per Hour	Per Cent. Solids in Feed
Marathon No 1	94 00	6 00	383 0	236.5	9 854	662 4	27 60	35.7
Marathon No 2	94 25	5.75	200.0	440 0	18 333	693.0	28.87	63.5
Hardinge	96 25	3.75	1,455 3	244 0	10.166	570.1	23 75	42.8
Chilean	93.75	6 25	1,401.7	237.0	9.875	951.4	39.64	24.9

Type of Mill	Ratio of Solids to Water	Water with Each Ton of Feed, Gallons	Water with 24-Hr. Tonnage, Gallons	Water in Pulp, Gallons per Minute	Kilowatts Consumed per Hour	Horsepower Consumed per Hour	Tons Dry Feed per Hp-hr.	Tons Dry Feed per Hp-day	Speed of Mills, R p m.
Mar. No 1	1 to 1 8	432.2	102,215	70 98	13.880	18.60	0.5316	12.758	30.27
Mar No. 2	1 to 0 57	138 0	60,720	42.17	16 791	22.50	0.8148	19.555	30.10
Hardinge	1 to 1.3	320.6	78,226	54 32	42.208	56 56	0 1797	4.313	29.03
Chilean	1 to 3.0	723.8	171,540	119.13	27.388	36.70	0.2691	6.485	39.88

The term pulp is here used to signify a mixture of water and dry feed, and the quantity of water in the pulp is the amount in combination with the dry feed.

The delays of the Hardinge mill are due to relining and allowing time for lining to "set." The delays of Chilean mill are due to renewing of crushing parts, replacing of screens and oiling of mill. The delays of the Marathon mill are due to changing of rod charge, addition of new rods, relining, and extraordinary repairs on defective parts.

The power consumed by the Marathon mill is practically constant, the rise in power consumed in test No. 2 being due to an increased charge of rods. The Hardinge power is also constant. The Chilean mill consumes less power when the tires and die are new than when they have been in operation for some time. If mills are slightly overloaded, the power increases very rapidly.

TABLE 3.—*Average of Test Conditions*

Type of Mill	Stationary Grinding Parts					Movable Grinding Parts					Total Grinding Parts
	Kind of Stationary Grinding Surface	Wt. of Stationary Grinding Surface, Pounds.	Life of Liners or Die, Days	Liners or Die Consumed in 24 Hr., Lb.	Liners or Die per Ton, Pounds	Charge Used in Mill	Wt. of Initial Charge, Pounds.	Total Pounds Added During Test	Pounds Consumed in 24 Hr	Consumed per Ton of Feed, Pounds.	Consumed Per Ton of Feed, Pounds.
Mar. No. 1 ..	Iron	4,480	82	54.707	0.2313	Iron	6,395	2,404	150.64	0 63697	0.86827
Mar. No. 2 .	plates	4,480	72	62.256	0.14149	rods	7,124	1,441	172.92	0 39295	0 53544
Hardinge..	Pebbles	6,740	159	41.761	0.17115	Pebbles	10,175	31,956	527.00	2 16	2.33115
Chilean	Die	1,722	63	27.333	0.11533	3 Tires	2,526	None	41 66	0.1762	0.29153

This table expresses the life of linings, the rate of wear, and pounds of each grinding medium sacrificed per ton of ore ground. The Chilean mill die corresponds to liners and lining of the other two mills. It is to be noted that the Chilean mill required the least amount of grinding medium per ton. It is the auxiliary parts, such as screens, screen frames, tire wheels or mullers, feed spouts, ploughs, mortars, and the power, that completely reverse this good quality.

TABLE 4.—*Comparison of Grinding Efficiencies*

Marathon No. 1						Marathon No. 2				
Mesh	Reciprocal of Average Diameter	Feed, Per Cent by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product	Mesh	Reciprocal of Average Diameter	Feed, Per Cent. by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product
+4	4.46	0 13	0.58	......	....	+4	4.46	0.09	0.40	......	....
+6	6.33	6.49	41.08	....	....	+6	6.33	9.37	59.31	0.04	0.25
+8	8.93	17.12	152.88	0.01	0.09	+8	8.93	21.58	192 71	0.81	7.23
+10	12.66	13.40	169.64	0.28	3.54	+10	12.66	15.90	201.29	4.90	62.03
+14	17.86	10 44	186.46	2 12	37.86	+14	17.86	13.55	242 00	12.05	215.21
+20	25.38	8 51	215.98	7.06	179.18	+20	25.38	11.14	282 73	15.55	394.66
+28	35.71	8.35	298.18	14.77	527 44	+28	35.71	10.42	372.10	16.82	600.64
+35	50.51	6.61	333.87	14.43	728.86	+35	50.51	6.96	351.55	12.50	631.37
+48	71 43	5.30	378.58	11.68	834.30	+48	71.43	3.92	280.01	8.53	609.30
+65	101.01	3.91	394.95	8.72	880.81	+65	101.01	1.97	198.99	5.43	548.48
+100	142.86	3.78	540.01	8.05	1,150.02	+100	142.86	1.15	164.29	4.67	667.16
+150	200.00	2 46	492.00	5.15	1,030.00	+150	200.00	0.66	132.00	2.82	564.00
+200	285 71	1 29	368.57	2.67	762.85	+200	285.71	0.37	105.71	1.70	485.71
−200	454.55	12.21	5,550 05	25.06	11,391.02	−200	454.55	2.92	1,327.29	14.18	6,445.52
Total ..	......	100.00	9,122.83	100.00	17,525.97	Total	......	100 00	3,910.38	100.00	11,231.56

TABLE. 4.—(*Continued*)

Hardinge Mill						Chilean Mill					
Mesh	Reciprocal of Average Diameter	Feed, Per Cent by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product	Mesh	Reciprocal of Average Diameter	Feed, Per Cent. by Weight	Relative Surface in Feed	Product, Per Cent. by Weight	Relative Surface in Product
+4	4.46	0.97	4.33	......	....	+4	4.46	0.94	4.19	......	....
+6	6.33	8.43	53.36	......	....	+6	6.33	7.71	48 80	......	....
+8	8.93	18.79	167.79	0.07	0 63	+8	8.93	16 49	147.26	0.05	0.45
+10	12.66	13 86	175 47	0.58	7.34	+10	12 66	11.84	149.89	0 10	1.26
+14	17.86	10 45	186 64	1.95	34.83	+14	17.86	8 98	160.38	1.07	19.11
+20	25 38	8 49	215 48	3.80	96.44	+20	25 38	7.18	182.23	3 63	92 13
+28	35.71	8 40	299 96	8.17	291 75	+28	35.71	7.43	265.33	7.75	276.75
+35	50.51	6 53	329.83	10.45	527 83	+35	50.51	6.04	305.08	9 08	458.63
+48	71.43	5 29	377.86	11.60	828.59	+48	71.43	4.80	342.86	9.38	670.01
+65	101.01	4.10	414.14	10.25	1,035 35	+65	101.01	3.89	392.93	8.53	861.62
+100	142.86	3.47	495.72	10.40	1,485.74	+100	142.86	3.76	537.15	8 95	1,278.60
+150	200.00	1.81	362.00	6.75	1,350 00	+150	200.00	2.58	516.00	6.23	1,246.00
+200	285 71	1.04	297.14	4.13	1,179.98	+200	285.71	1.82	519.99	4.21	1,202.84
−200	454.55	8 37	3,804.58	31.85	14,477.42	−200	454.55	16.54	7,518.26	41.02	18,645.64
Total ...	......	100.00	7,184.30	100.00	21,315.90	Total		100 00	11,090 35	100.00	24,753.04

	Marathon No 1	Marathon No. 2	Hardinge	Chilean
Work units in product	17,525 97	11,231.56	21,315 90	24,753 04
Work units in feed	9,122.83	3,910.38	7,184.30	11,090.35
Work units expended in crushing	8,403.14	7,321.18	14,131.60	13.662.69
Work units expended in crushing corrected for tonnage.........	107,207.3	143,165.70	60,949.60	88,602.54

Excess Efficiency of Marathon No. 2 over	Per Cent.
Marathon No. 1	33.54
Hardinge..................................	134.80
Chilean	61.58
Excess Efficiency of Marathon No. 1 over	**Per Cent.**
Hardinge..................................	75.90
Chilean	30.52
Excess Efficiency of Chilean over	**Per Cent.**
Hardinge..................................	45.37

Tyler Standard screens were used in all of the screen analyses, and constants of this screen scale are employed in computations. The principle upon which the comparison of efficiencies is based may be summed up in a few words: "Work done in crushing is proportional to surface exposed in crushing, and therefore nearly proportional to reduction in diameter," or, "nearly proportional to the reciprocals of diameters crushed to." This is the well-known and substantial efficiency comparison of Del Mar. Per cent. weight is multiplied by reciprocals of *average diameter* in both feed and product, the difference in these two quantities measuring the work expended in crushing. When this quantity is multiplied by tons per horsepower-day, we involve the two factors that complete the efficiency comparison. This method is commendable with but one exception; the assuming of a value for the average diameter of −200-mesh material. This assumed value probably comes somewhere near to the approximate value, but no more. In logarithmic plotting of feeds and products, Fig. 6, the lines have been extended in a smooth curve to the edge of the paper, and in every case the assumed value is only approximate. It is of no advantage to assume this value, especially when there is large quantity of −200-mesh material. The curve of Chilean product lacks sufficient points to tell just where it does extend. The others have a fair degree of accuracy.

Crushing-Surface Diagram

This is the method proposed by Arthur O. Gates for comparing efficiencies in grinding. Cumulative percentages are plotted vertically and reciprocals of screen-mesh diameters, horizontally (Fig. 5). This

TABLE 5.—*Gates Method of Comparison*

Mesh	Reciprocal of Mesh Diameter	Mar. No 1 Feed Cumulative, Per Cent by Weight	Mar. No 1 Product Cumulative, Per Cent by Weight	Mar No. 2 Feed Cumulative, Per Cent by Weight	Mar. No. 2 Product Cumulative, Per Cent. by Weight	Hardinge Feed Cumulative, Per Cent. by Weight	Hardinge Product Cumulative, Per Cent. by Weight	Chilean Feed Cumulative, Per Cent. by Weight	Chilean Product Cumulative, Per Cent. by Weight
+4	5 40	100.00	....	100.00	....	100.00	.. . ..	100 00	....
+6	7.63	99 87	....	99.91	100 00	99.03	.. ....	99.06	......
8	10.75	93.38	100 00	90.54	99.96	90.60	100 00	91.35	100.00
10	15 38	76.26	99 99	68.96	99.15	71 81	99.93	74.86	99.95
14	21.74	62.86	99.71	53.06	94.25	57.95	99.35	63.02	99.85
20	30 48	52.42	97.59	39.51	82.20	47.50	97.40	54.04	98.78
28	43.10	43 91	90.53	28.37	66.65	39.01	93.60	46.86	95.15
35	60 98	35 56	75.76	17.95	49 83	30.61	85.43	39.43	87.40
48	86.20	28.95	61 33	10.99	37.33	24 08	74 98	33.39	78 32
65	121.95	23 65	49 65	7 07	28.80	18 79	63.38	28.59	68.94
100	172.41	19 74	49.93	5.10	23 37	14.69	53.13	24.70	60 41
150	243.90	15.96	32 88	3.95	18.70	11.22	42.73	20.90	51.46
200	344.82	13.50	27.73	3 29	15 88	9.41	35.98	18.36	45.23
−200	....	12 21	25.06	2.92	14.18	8.37	31.85	16.54	41.02

TABLE 5.—(*Continued*)

From Crushing Diagram by Gates Method (Fig 5)

	Square Inches	Efficiency Corrected for Tonnage
Marathon No. 1	7.51	95.812
Marathon No. 2	6 65	130.004
Hardinge.	12.02	51.842
Chilean ..	11.12	72.113

Excess Efficiency of Marathon No 2 over	Per Cent.
Marathon No. 1	35 69
Hardinge..	150.75
Chilean	80.29
Excess Efficiency of Marathon No. 1 over	**Per Cent.**
Hardinge..	84.81
Chilean	32.86
Excess Efficiency of Chilean over	**Per Cent.**
Hardinge	39 10

diagram averages the diameters without calculation, and areas upon it are proportional to surface produced, and, in accordance with Rittinger's law, to energy spent in crushing alone. Measuring these areas up to 200 mesh and multiplying by tons per horsepower-day, we obtain efficiency units for comparison, as indicated in Table 5.

TABLE 6.—*Comparison of Del Mar and Gates Methods*

Excess Units by	Del Mar, Per Cent.	Gates, Per Cent.	Numerical Average, Per Cent.
Marathon No. 2 over			
Marathon No 1.	33.54	35.69	34.61
Hardinge.	134.80	150.75	142.77
Chilean ..	61.58	80.29	70.93
Marathon No. 1 over			
Hardinge.	75.90	84.81	80.35
Chilean ...	30.52	32.86	31.69
Chilean over			
Hardinge.	45.37	39.10	42.23

In view of the fact that in Del Mar's system the −200 mesh material is taken into account and in the other it is not, we may consider a check existing between the two methods with but one exception, comparing

Marathon No. 2 with Chilean, the two extremes of feeds and products; the one producing the most slime and the other producing the least slime. To be impartial and conservative at the same time, we will adopt the numerical average of the two methods.

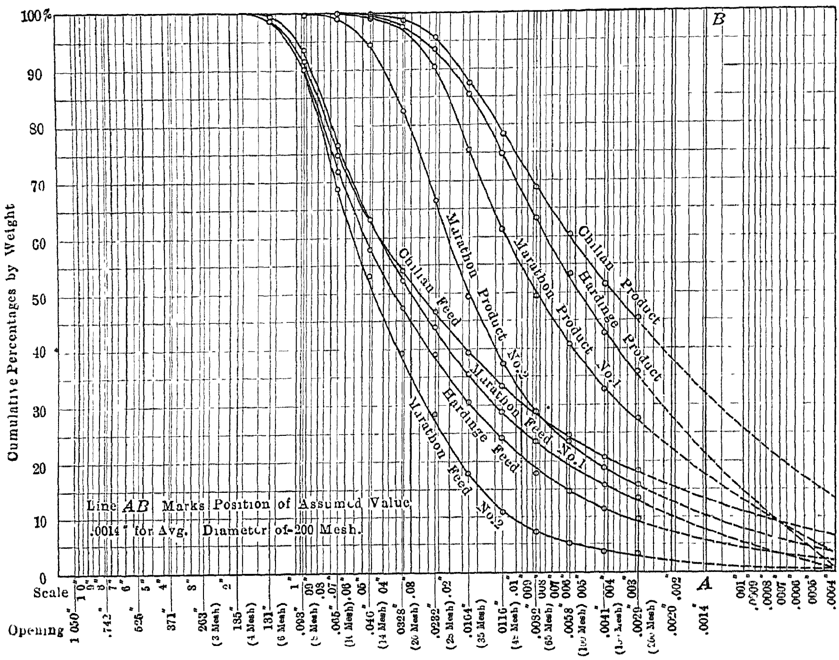

FIG. 6.—CUMULATIVE LOGARITHMIC PLOT OF SCREEN ANALYSES ON FEEDS AND PRODUCTS.

SUMMARY

For the material treated and under the conditions of these tests, the Marathon mill is far superior to the Hardinge and Chilean mills in grinding efficiency. Of the Hardinge and Chilean mills, the Chilean is superior to the Hardinge. The Marathon mill will bear further investigation and experiment. Its costs will be much less in the actual installation of several mills than in a single mill requiring perfection of parts, as did this one.

On the basis of the results obtained in these tests, two mills, 4 ft. in diameter by 8 ft. long, for both coarse and fine grinding (Fig. 7) have been ordered, the principal features of which are as follows:

(*a*) Feed scoop attached to the head end of the mill. There are three channels in this scoop, entrance of each channel having a rectangular area of 108 sq. in. (9 by 12 in.).

(*b*) Sectional liners for head end and solid circular liner for tail end.

(*c*) Rigid base, with no tilting arrangement.

(*d*) Solid roller bearings (not spiral).

(*e*) Driving gear attached to tail end of shell, the pulley being supported rigidly by having its bearings set solidly on a separate concrete

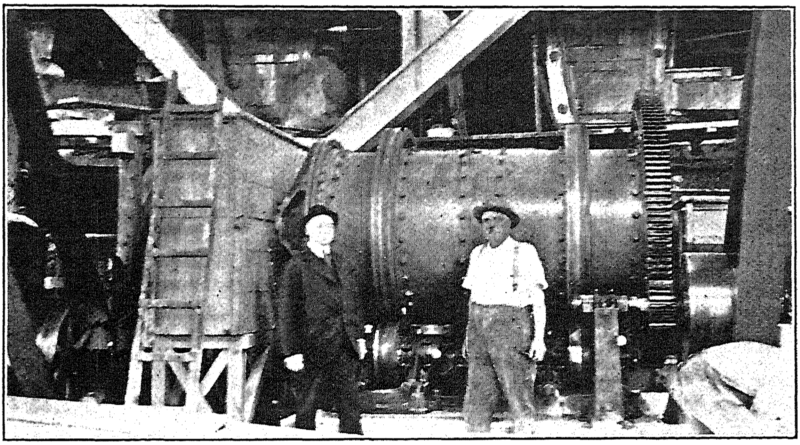

FIG. 7.—NEW MARATHON MILL.

base. One of these mills has been in operation for a few days and is running smoothly after having considerable trouble with a hot bearing. These bearings are not able to stand up under such service. Babbitted bearings will undoubtedly remedy this trouble.

Thanks are due to Milton H. McLean, General Manager of the Detroit Copper Mining Co. of Arizona, for permission to assemble and publish the above data.

DISCUSSION

THE CHAIRMAN (B. B. GOTTSBERGER, Miami, Ariz.).—On your trip today through the Inspiration and Miami mills you have seen in actual operation the machines which represent the changes adopted in grinding methods in the Miami district from the beginning of operations in March, 1911. In one section of the Miami plant are three Chilean mills not yet discarded, representing the first machines adopted. In another section, pebble grinding is still in operation, while in the balance of the plant, pebbles have been replaced with iron balls. In all cases these machines are handling a feed which is the product of roll grinding down to about ¼ in. in size. The latest step in grinding in the district, you saw at the Inspiration plant, where in Marcy mills a feed, the larger pieces of which remain on a 2-in. opening, is being reduced in one opera-

tion to a product of which only about 3 per cent. remains on the 48-mesh screen. A detailed comparison of the methods used at the two plants would be very instructive but our own work at Miami has not yet reached a point where such a comparison is possible, the reason being that our final product is not yet fine enough. By means of larger Hardinge mills operating in closed circuit with Dorr classifiers soon to be installed in one section of the plant, we hope to obtain under properly balanced operating conditions the same end result as at Inspiration. It might interest you, however, to know some of the results we have obtained here by the substitution of balls for pebbles. With pebble grinding, using three 8-ft. by 22-in. mills, followed by one 8-ft. by 66-in. mill for regrinding the coarse product of the primary mills, we handled 636 tons per 24 hr. in one section of the plant. Substituting balls for pebbles in the first operation and using two mills in place of three we obtained a tonnage of 714 tons to the section. With the use of balls both in the primary mills and in the regrinding mills, the tonnage for one section was raised to 822 tons per day. Coincident with this increase in tonnage, we also obtained a greater efficiency. The use of pebbles in all mills showed 21.5 per cent. plus 48-mesh material in the final tailing, which figure was reduced to 13.9 per cent. for balls followed by pebbles, and to 4.6 per cent. with the use of balls both in the primary and regrinding mills.

We should be able to have an interesting discussion on this subject this afternoon if some of the mill men present will participate, particularly as we have before us Mr. Blickensderfer's paper, presenting what may be one step more in the evolution of fine grinding.

R. B. Yerxa, Miami, Ariz.—I would like to ask if the Marathon mill can operate at such a tonnage as to produce the same product in fineness as that produced by the Hardinge and Chilean mills?

F. C. Blickensderfer.—At the time this paper was published, such had not been the case. However, the Marathon mills of the larger type, now in operation at the Burro Mountain Copper Co., are grinding considerably finer than any product mentioned in this paper. Those mills at Tyrone are, roughly speaking, operating on 300 dry tons per 24 hr., and are grinding this down from 15 to 18 per cent. plus 48. The power consumed is about 48 and the energy units corrected for tonnage are 130,000.

A. P. Watt, St. Francois, Mo.—I was much interested in reading Mr. Blickensderfer's paper comparing the efficiency of the Chilean, Hardinge and Marathon mills. The question of fine grinding is now receiving considerable attention in southeast Missouri. At present the jig middling, which is through 10-mm. on 2-mm., is generally crushed in

rolls. One company, however, is using a Chilean mill and another company is using a Huntington mill for this purpose. Recently, however, the use of pebble and ball mills has been considered for solving the problem of fine grinding. Hardinge and Marcy mills are in regular operation in the district for the grinding of middling and two companies are to install Marathon mills for the same purpose.

By reason of our grinding problem I was much interested in Mr. Blickensderfer's paper, but as I read it a few questions arose in my mind. One was that in the test the Marathon was using metallic rods and was being compared against the Hardinge which was using pebbles. It is well known that the use of steel pebbles—so termed—in the Hardinge mill increases its efficiency. I wondered if that fact had been given consideration in later tests by running the Marathon against the Hardinge, using steel pebbles instead of flint pebbles in the Hardinge. I noticed that the Marathon feed contained about 63 per cent. solids while the Hardinge mill feed contained but 40 per cent. Would not an increase in the percentage of solids in the feed to the Hardinge increase the efficiency of that machine?

I would like to ask as a matter of information whether the rods in the Marathon mill still retain their cylindrical shape after continual use or if they become elliptical? Also, do the rods retain the same diameter at both ends after use, or do the rods at the feed end wear more than at the discharge end?

F. C. Blickensderfer.—In answer to the first question, as to whether or not we have tried steel balls in the Hardinge mill, I will say, "no." The idea of using steel balls in cylindrical grinding machines was just being introduced. The second question regarding the wear of rods is one which I failed to make sufficiently clear. The rods used in the small Marathon mill of 7-ft. length, wore almost uniformly from end to end until they reached a diameter of ½ in. After that some of them would flatten out, take an elliptical shape, and all rods would wear with a slight tapering toward the feed end, due no doubt to the fact that the particles of coarse feed kept the rods at a greater distance apart in the region of the head end than at the discharge end, so that the surfaces of the rods at the head end were constantly exposed to rock. The tail end was occupied by finer particles of pulp. Lately, in our present practice, due to abnormal conditions regarding the supply of steel, we have not obtained successful results by using the present supply of rods. This, I think, is due to poor steel. We have had rods 1 in. in diameter break in two. We have had rods nearly ½ in. in diameter go into the shape of an ellipse, and then again we have had defective castings. These are mechanical defects that are well on the way to perfection, and will not seriously retard the ultimate success of the mill.

R. B. T. Kiliani, New York, N. Y.—Mr. Blickensderfer's paper is of considerable interest as it gives the first published and authoritative data on the operation and performance of the Marathon mill and to this extent it may be considered an interesting contribution to our knowledge of milling machinery. No data are given as to the metallurgical results obtained with the different machines, and to this extent, it is not conclusive as to their relative suitability for the work under consideration. The extraction and grade of concentrate obtained by the use of various types of grinding equipment are naturally of at least equal importance to their actual mechanical efficiency. In other words, the net profit per ton of ore handled is the final criterion of economic efficiency. The mills were not operated at the same time, and no attempt was made to vary the conditions of operation to obtain the maximum efficiency from each, except possibly in the case of the Marathon mill. For these reasons, it is not a "comparative test," although, as above mentioned, thanks are due the author for some interesting data.

The Chilean and Hardinge mills were operated together for a period of 63 days, but after the first 30 days, it became necessary to shut down the Hardinge for relining, as it was started with an old lining. Owing to the type of lining used, flint pebbles set in cement, a longer time should have been allowed for this work, and especially for the cement to set, than the 2.36 days stated by the author. The short life of 159 days for the lining of the Hardinge mill may be ascribed to this fact, as well as other conditions which will be taken up later. The actual percentage of time lost through all delays in the operation of Hardinge mills at other plants is well under 1.5 per cent., when proper care is taken that they operate under conditions which have been found to be best. The life of the same type of lining at the plant of the Arizona Copper Co., immediately adjoining the Detroit Copper Mining Co., is as high as 400 days, with an average life of over 200 days.

Two other conditions to which may be ascribed the short life of the lining are the dilution of the pulp and the speed at which the mill is operated. It is a well-known fact that for most efficient operation of a pebble mill, the solids in the pulp should be in the neighborhood of 60 to 70 per cent., and under these conditions, the wear of the lining is very much reduced. The speed of the mill of 29 r.p.m., mentioned by the author, is too high for this type of lining, since the action of the pebbles is such that they are thrown across the mill and strike the lining instead of acting upon the pulp. This, of course, causes the lining to wear out faster than would otherwise be the case. These two conditions of pulp consistency and speed of the Hardinge mill also affect the consumption of pebbles, which is very considerably higher than that at the No. 6 concentrator of the Arizona Copper Co., which was 1.334 lb. per ton

ground, for the year 1914, as against the figure of 2.33 lb., as given in the paper under discussion.

More important, however, than the effect upon consumption of lining and grinding mediums of these two factors of speed and pulp dilution, are their effect upon the capacity obtained from the mill. It has been found that a dilute pulp with a high percentage of moisture, tends to very much decreased capacity and somewhat higher power. Curves and other data showing the effect of pulp dilution on capacity, fineness of grinding, efficiency, and power have been published for the past 7 or 8 years, and it would therefore be merely a repetition to give these in this discussion. Suffice it to say, however, that the efficiency of pebble mills has been increased over 50 per cent. by thickening the pulp fed to them. Had experiments been made to ascertain the proper speed and pulp density, the results with the Hardinge mill would have been much more satisfactory.

As already mentioned, no information is given as to the metallurgical results obtained with the different machines, other than the statement that the total recovery after the Marathon mill "was almost exactly the same" as with the Chilean mills. It is a well-known fact that the recovery by gravity concentration when grinding in Chilean mills is lower than when the ore is ground in pebble mills of the conical type. This does not necessarily hold true when flotation follows gravity concentration, but it is found that the grade of the concentrate is lower, and that it contains a higher percentage of insoluble matter, when grinding in Chilean mills. The Marathon-mill discharge, however, owing to the fact that it does not contain very much minus 48-mesh material, requires regrinding in some other type of machine in order to obtain the extraction and grade of concentrate resulting when grinding the same ore in the Hardinge mill. By referring to the author's data, we find that a considerably greater proportion of the Hardinge-mill product is finished than is the case with that of the Marathon mill.

Mention is made in the author's paper of the fact that metal pebbles are being used in place of flint for grinding in Hardinge mills, and the question arises why these were not tried at the plant under consideration, as it is a known fact that the efficiency of the machine is increased by their use, provided the same volume is used as of flint pebbles. It has also been found that they are cheaper than flint, considering all items of expense, except in certain exceptional cases. Another point in favor of their use, is the fact that increased recovery can be obtained without the expected higher grinding cost. Another fact which might be mentioned at this point is that operating the mill in closed circuit with a Dorr, or other similar classifier, increases the efficiency and therefore decreases the grinding cost. It might be claimed that a similar increase in efficiency would be obtained by operating the Marathon mill in a similar manner, but this, to the writer's mind, is doubtful, owing to the much

greater circulating load, due to the fact that this machine is unable to grind fine, except in larger machines where the capacity drops and the power increases very rapidly.

For a true comparative test, both machines should be operated side by side, taking the same feed and delivering the same final product, and the metallurgical results obtained with each taken into consideration. The Hardinge mill should also be operated with a metal lining and should use metal pebbles, since it would then be using the same class of grinding mediums as its competitor. It would naturally be absurd to suggest the use of a flint lining and flint rods in the Marathon mill, but the other has been proven to be entirely practicable. Should such a test be made, it would be interesting to compare the results with those reported in Mr. Blickensderfer's paper.

F. C. BLICKENSDERFER.—It is mentioned in this paper that the total unit tonnage of 440 tons per 24 hr. was sent through the small Marathon mill. On the other unit we had three Chileans in operation. This test extended over 4 days, so we were reasonably safe in drawing our conclusions. There was a difference of less than 1 per cent. in the extraction of the two units, operating as I have outlined to you. It has always been our custom to assay the mesh sizes of all screen analyses made in the concentrator. I am not able to answer the question as to what condition the material on the screen sizes was in for concentration. There are no accurate methods for separating mineral from gangue in exactly the same manner as the gravity machines do. The nearest thing I know of is by means of a heavy solution of mercuric iodide and potassium iodide, with a specific gravity of about 3.20. That portion of ore which sinks is concentrates; that which floats is tailings. There is a very definite separatory action in this method. In incorporating this method of separating the concentratable material from the gangue in the screen sizes, much difficulty was encountered, especially in treating slimes, so that, to my knowledge at present it is an open question as to the determination of the free mineral in the screen sizes.

The Hardinge mill slimed slightly less of the total material fed to it; the Hardinge mill slimed a little more of the mineral fed to it, as compared with the Chilean and Marathon. The method of determining to what extent the mill slimes mineral was as follows: All of the material in the feed which remains on a 200-mesh screen was assayed and its copper content obtained by multiplying per cent. weight by per cent. copper. Then all of the material in the product which remains on a 200-mesh screen was assayed and its copper content similarly obtained. Subtracting the copper content thus obtained in the tails from the copper content in the heads gives the copper content of the material actually slimed in the process of grinding.

Proof of Statement that Hardinge Mill Slimes More Mineral than Chilean

Assume Weight of 100 Tons

	Tons	Per Cent. Copper	Tons Copper
Total + 200-mesh pulp in Hardinge feed......	91 63	1.03	0.944
Total + 200-mesh pulp in Hardinge product....	68 15	1 01	0 688
Material actually crushed to −200-mesh......	23.48	1.09	0.256

Assay of material actually crushed to −200 = 1.09 per cent. copper. And in case of Chilean mill:

	Tons	Per Cent Copper	Tons Copper
Total + 200-mesh pulp in Chilean feed.......	83.46	1.15	0.960
Total + 200-mesh pulp in Chilean product....	58 98	1.19	0 702
Material actually crushed to −200-mesh......	24 48	1.05	0 258

Assay of material actually crushed to −200 = 1.05 per cent. copper.

Hence, the Hardinge mill has slimed more mineral to produce an assay of 1.09 per cent. copper than the Chilean in producing an assay of 1.05 per cent.

S. J. Jennings, New York, N. Y.—I would like to ask Mr. Blickensderfer a question. In his explanation of the action of the Marathon mill, he shows a diagram which does not seem to differentiate between the two possible actions that take place in the mill—one is a crushing action and the other is a grinding action. If we rotate a cylindrical mill somewhat rapidly so that the pebbles are entrained on the up side and fall, we would have a crushing action. If the rapidity of rotation is not sufficiently great to entrain the pebbles on that side but merely to raise them up sufficiently and allow them to go back, you have a continuous grinding action. I would like to ask Mr. Blickensderfer if the Marathon mill was run at such a speed as to get a crushing action, or a grinding action. I assume from the fact that the rods wear evenly, with the small difference he has noted, he has merely a grinding action in his mill; but I would like to know that definitely.

F. C. Blickensderfer.—We have going on in the Marathon mill both a grinding and a crushing action. In the explanation referred to, I purposely inserted the words "strike" and "rolling together." Of the charge of rods, when observed inside the revolving mill, about one-third is flying through the air and the other two-thirds are exercising a rolling action on one another. This is better illustrated by standing near a Marathon mill and hearing the constant striking of rods as they leave the periphery and travel across to the opposite part of the mill.

R. S. Handy, Kellogg, Idaho.—In the Marathon mill, in the matter of keeping them parallel—when the rods wore to small diameters, was there any difficulty in the rods getting crossed and not maintaining parallel relationship?

F. C. Blickensderfer.—We found that about 8 days' time was the limit that these mills could run without having the rods changed. The rods in the 8 by 12 mill catch hold of something and they twist up and form all kinds of grotesque shapes; it is time to stop the mill. The operator should keep close watch and remove the small twisted rods whenever a few appear in the top of the charge. In most cases, we straighten them out and put them back in again. The smaller ones are thrown away, and the load of rods brought back to about 7 tons, which is determined by filling the mill exactly one-half full of rods.

R. B. Yerxa.—I would like to ask what the tonnage was on the operation with Chilean mills when the test was made to find out what the recovery was between the two different methods of grinding.

F. C. Blickensderfer.—The tonnage was approximately the same. We had plunger-type feeders which governed the tonnage by the distance the gate was opened. We carried those gates at the same height, and the mill bins being filled at the same time, the tonnage of the two units would be approximately the same—within 5 per cent., I would say.

The Chairman.—I think we all ought to be grateful to Mr. Blickensderfer, in the first place for this paper, and again for his very full answers to our questions. I think that at times when we get to discussing a question of this kind, we are apt to give the impression that we may be criticizing, but I hope Mr. Blickensderfer does not feel that way in this case, as I believe he has brought something forward that is going to make us think. Present-day methods of concentration for handling the so-called disseminated copper ores call for fine grinding and it will only be by comparison of results obtained at different plants that progress will be made.

Referring to the results in the present paper, I am impressed by the fact that taking −48-mesh product as the desired result, the Marathon mill yields a comparatively coarse product—one which in our case would not be suitable for the work we have to do. If Mr. Blickensderfer would not mind answering one more question, I would like to ask if he thinks putting the mills in closed circuit with drag classifiers would help in the fineness of the product.

F. C. Blickensderfer.—In the efficiency comparison of these machines, I call your attention to the fact that the Marathon mill when running with a tonnage of 440 turned out an efficiency, corrected for tonnage, of 143,000. When it was running as a finishing mill, handling 235 tons in 24 hr., its efficiency was 107,000. Now, I take it from those two figures that if the Marathon mill was filled up, so to speak, with coarse material from its own discharge, much better results could be obtained. I have never seen any figures on those mills grinding in closed circuit.

C. W. Merrill, San Francisco, Cal.—I am not going to ask Mr. Blickensderfer a question. I am going to congratulate him. We men who have been studying the evolution of fine grinding in connection with gold milling for the past 20 years have always rebelled a little at the spherical contact—the point contact that we have had to contend with in all of the ball mills. We have felt that it was an extravagance of power that eventually would be eliminated; and when we prepared our paper for the International Engineering Congress on the subject of fine grinding, we ventured to predict that the knowledge we had of the small Marathon mill would result in just such work as Mr. Blickensderfer has explained to us today. I think there is a new principle involved in this machine—that of linear contact, as compared with spherical contact—and I want to express my appreciation of the work Mr. Blickensderfer has done.

F. C. Blickensderfer.—I should like to thank the Institute as a whole, and the members, individually, who have been so kind as to bring up these points. They seem to have brought out a better understanding regarding this mill. In publishing this paper, I wanted to be sure that we were right in drawing our conclusions concerning the Marathon mill. I feel that everything which has been said has been corrected in so far as it was possible to correct it. It might be of passing interest to note that every sample taken from the mill, all screen analyses, and all meter readings were made by the writer. It is not for the Marathon mill alone that I stand, but it is for the principle of the grinding element which the gentleman has just mentioned; and I think, and predict with equal emphasis, that within another 5 years your ideas on the subject of fine-grinding machinery will have a different viewpoint.

Robert Franke, Miami, Ariz.—I would like to make a few remarks in regard to the use of equating the work performed by crushing. The tendency of this use has been rather noticeable during the last few years, and in this discussion has digressed to stating the capacities of grinding machines in terms of "energy units." I realize the need of resorting to an equating medium for efficiency comparisons, and favor its practice, but feel that we should be cautious, in view of several unknown and disputed factors, to apply this principle only within its limits of reasonably known accuracy, or under very analogous conditions. Otherwise, erroneous deductions may be the result.

The factors to be regarded are these: Rock is a heterogeneous substance, and in the porphyries generally is composed of feldspars bonded by a siliceous matrix. As crushing proceeds from a coarser to a finer state the softer components are first reduced to colloidal sizes until practically only the harder components remain for final crushing. Hence

there exists an increasing unit resistance to crushing for which a determined allowance should be made in equating the production units.

Rittinger's and Kick's laws, upon which the interpretation of work performed in crushing is based, are in dispute, because of a disagreement as to the manner in which crushing forces act. Relatively, these two laws are fairly parallel for the coarser range of sizes, but become very divergent with the finer sizes.

The average size of −200 material is an unknown quantity, and in our present knowledge thereof can only be unreliably approximated. Moreover, it is also to be kept in mind that with this range of sizes there are encountered two classes of material, sands and colloids, each of which undoubtedly have different surface-volume relations.

In view of these factors, it is my opinion that the conversion of the work performed by crushing into energy units, for comparison of efficiency, should only be attempted when the fines of the feeds and products to be compared are practically equal, say within 2 per cent. of each other. Further, the practice of stating the capacities of grinding machines in terms of "energy units" should not be adopted until the absolute value of these units has been determined, or misunderstandings will result. In lieu thereof, the practice of stating such capacities in terms of "Tons through a given mesh per horsepower unit" (which mostly is the primary object aimed for), for a given ore and given feed, would be more reliable and be resorted to until research along these lines will broaden the field for more exact application.

F. J. H. Merrill, Los Angeles, Cal.—I would like to ask if the manufacturers of grinding balls have in any way made a suggestion in providing sections of rods or in casting balls with spherical cavities in the hope that the balls will engage the adjacent one. I think the Jeffrey company makes balls of that kind.

Theodore B. Counselman, Duluth, Minn. (communication to the Secretary*).—I have read Mr. Blickensderfer's paper with special interest because at the time these tests were being carried on I occupied the position of Efficiency Engineer at No. 6 concentrator of the Arizona Copper Co., Ltd.

A comparison of the work of the 8-ft. by 36-in. Hardinge pebble mills, at the neighboring concentrator of the Arizona Copper Co., Ltd., immediately suggests itself. This plant contained 12 mills exactly the same as the mill used in this test. In Table 1, the figures for this plant are averages for the year 1914.

The feed to the mills at No. 6 concentrator, was deslimed in drag belts. The tonnage is low because it was not necessary to crowd the mills. Ten would have been, and frequently were, sufficient for the

* Received Oct. 7, 1916.

TABLE 1

	Detroit Copper Co.	Arizona Copper Co
Power, horsepower	56.56	56 65
Tonnage	244.00	196.70
Dilution, per cent. solids	42 80	55 00
Pebble consumption per day	527 00	262 00
Pebble consumption per ton	2.160	1 334
Life of pebble lining	159.00	200.00
Tons per lining		40,000 00

crude ore tonnage at that time being handled. When the mills were first installed, 500 lb. pebbles per day were charged for an entire month, without seeming to create an excess. This amount was gradually cut down to 250 lb. where it remained.

The pebble lining used in all these mills was first introduced by David Cole, then in charge of the remodeling of No. 6 concentrator. It consisted of the largest-size Danish pebbles, set on end in neat cement or very rich concrete, the pebbles being rather carefully fitted together, much in the manner of building a dry-wall. The lower half of the mill was lined the first day. The mill was turned over the second day and the other half lined. It could then be put into operation, if desired, but was ordinarily allowed a day or two longer to take a final set. Such a lining required 10,000 lb. of pebbles, 50 sacks of cement, and cost installed, including cutting out the old lining, about $160. The life would vary from 150 days to as long as 400 days actual running time, with an average of about 200 days. Linings composed of silex blocks would last only about 60 days.

Lifting bars were tried in these mills. These were formed by rows of silex blocks, set on end, in the cylindrical portion of the mill. Repeated tests could detect no advantage from them.

A curious fact was brought out by several tests to determine the best dilution of the pulp. Mr. Blickensderfer mentions the "accidental oversize" in the discharge of the mill. The same thing was noticed at No. 6 concentrator, and the mill operators would correct it by "sticking a hose in the feed launder." This seemed so contrary to accepted ideas, that carefully conducted tests were made between several different pairs of mills, each pair having feed from the same source, but water being added to one while the feed to the other was as dry as possible. In repeated tests conducted with the utmost care, it was shown that with greater dilution, there would be less accidental oversize. It was decided at that time that the most satisfactory way to operate these mills would be in closed circuit.

The Marathon mill at the West Yankie Concentrator was of great interest to all of us. The low power consumption was, of course, the direct result of the small diameter of the mill. I very much doubt if the

same attractive difference in the power consumption of the Marathon and the Hardinge mills, would hold true with a larger-sized Marathon designed for finer grinding. In fact, this is shown by figures from Burro Mountain.

The capacity and general character of the work of this mill seems to be explained by the fact that its action is that of a series of long-faced rolls. The wide face precludes the possibility of accidental oversize getting through, and it would be conceivably possible to pass an enormous tonnage through the machine, if no questions were asked as to the resulting grinding. The Marathon mill, judging from results given in Mr. Blickensderfer's paper, produces very little slime. This was noted all through the tests on the Marathon mill. Had this machine been invented 10 years before flotation was perfected, it would have solved many a difficult milling problem. Now, however, copper metallurgy has so changed that it is advantageous to make as much slime (minus 200 mesh) as can economically be made at each grinding. In 1913–14 the Miami Copper Co. replaced their worn-out Chilean mills with Hardinge pebble mills in order to get a more granular product. Now with flotation installed they are using balls instead of pebbles and grinding in closed circuit, in the endeavor to make more slime.

In comparing the results of the Marathon and Hardinge mills in these tests, the fact should not be overlooked that the Marathon mill was equipped with steel rods, while the Hardinge mill used flint pebbles. It would be ludicrous to suggest that the Marathon be equipped with flint rods, but to suggest that iron balls be tried in the Hardinge would be quite to the point. In one case that I know of the capacity of the mill was increased more than 60 per cent., and the fineness of grinding increased by substituting balls for pebbles.

The measurement of grinding efficiencies has received a good deal of attention, and two well-known theories and methods have been advanced, of one of which, and a modification of it, Mr. Blickensderfer makes use in his paper. The other would give a very comparable result. To my mind, however, both methods fall down in that ore is not a homogeneous substance. The ores of the Morenci district, for instance, are relatively easy to granulate to, say, 20 mesh, but very much more difficult to pulverize to 200 mesh. The only real comparison between grinding machines would be to equip otherwise identical sections of a mill with the two machines to be tested, and determine the net profit made by each section. In one case that I know of, balls versus pebbles in Hardinge mills have been compared in this way, with the advantage very much in favor of the ball-mill section, despite the fact that the relative mechanical efficiency was only slightly better.

I do not think that the Hardinge mill has been really tried out in Mr. Blickensderfer's tests. It should have been operated with the steel lining and the steel balls, and working in closed circuit with a Dorr, or

similar classifier. Had both mills been equipped in this way, and operated to give products of equal fineness, I think there would have been little to choose between them. I should like very much to see this comparison made.

The Marathon mill should have a field where slime is anathema, where the mineral can and should be saved at a relatively coarse size. As Mr. Blickensderfer points out, there were several mechanical defects in the first machine, which have to some extent been corrected in the second. These defects can doubtless be eliminated entirely, if the machine proves to have a wide enough application.

The decline in the popularity of Chilean mills is due more to mechanical drawbacks than to quality of work. The wear of tires and dies is not serious, but as Mr. Blickensderfer himself notes, it is the auxiliary parts, the screens, screen frames, mullers, feed spouts, ploughs, etc., that give the trouble, not only in repairs but in operation.

David Cole, El Paso, Texas (communication to the Secretary*).—The paper by Mr. Blickensderfer presented at the session at Miami was very interesting to me because I witnessed the operation of the machines when they were being tested. I can vouch for the great pains that were taken by Mr. Hall and Mr. Blickensderfer in making the tests as nearly parallel as possible as to feed going to the competing machines, the power consumed by them, and the measuring of results as accurately as possible. Mr. Blickensderfer stated in his discussion that the tables compiled by him were done with great care and "are about as accurate as they could well be," and I have pleasure in corroborating his statement from the standpoint of observation on the ground at the time. We are indebted to Mr. Blickensderfer not only for his paper, but also for the very courageous way in which he defended the position he had taken, when under the rapid fire of questions that resulted.

The errors of comparison are fundamental and the question raised by Mr. Watt discloses one of the most important ones. Except for the purpose of showing different results when using grinding media of different gravity, it is fundamentally wrong and illogical to compare the work of a grinding mill of the tube-mill family, using flints, with one using steel grinding media. It would be illogical to suggest that the Marathon mill be operated with a set of rods made of agate, and it is equally illogical to compare the work of a Hardinge mill using flints with a Marathon using steel rods. Little importance should therefore attach to the comparison.

The comparison of work done, based upon the scientific theory of Stadler, Gates, Kick, et al., is beautiful on paper, but there are a lot of us who hesitate to accept the theory as "law." We are inclined to

*Received Oct. 4, 1916.

regard a direct comparison of grinders arranged side by side, getting feed from a common source through a mechanical distributor, and making a product that affords as nearly as may be the same screen measure, and at any rate affording an equal metallurgical opportunity for the subsequent treatment, as the Supreme Court in these grinding matters. The Marcy versus Hardinge ball-mill controversy is soon to have this kind of a hearing at the Inspiration plant, and the results will be watched with great interest, and if Mr. Blickensderfer can arrange to have the Hardinge mill at Morenci blocked down to about 6 ft. in maximum diameter, have it lined with steel with lifter bars, and filled with a charge of properly assorted steel balls, and then repeat the comparisons mentioned in his paper, our knowledge of fine grinding will be much advanced.

I believe in the use of the rods. I feel that the line of contact affords a better opportunity for the power being expended to be applied to the particles to be broken or crushed. I believe that the large-ball idea will prove the better scheme for breaking down the coarse particles, say to 8-mesh size, but that the rod idea will win out for the finishing process, particularly in closed circuit with an overflow classifier. But I do not think that the difference will be so overwhelming as to put the ball type in the discard, so to speak, for the balls may be modified in shape so as to get the line of contact result. What will prove to be the best length of rod, or the proportion of length to diameter, have not been determined. Perhaps a length equal to a single diameter, but with a dual axis will prove to be good. A "grinder" of this form presents a circular side and end elevation and a square appearance in plan. It is balanced around its center of mass similarly to a sphere and affords lines of contact of lengths varying from nil to the largest diameter. It would, of course, tend to wear into a sphere, but it ought to do a lot of grinding before it finally surrenders to that form.

There is one point distinctly in favor of rods, and that is that the steel mills everywhere are equipped to produce them at a minimum expense. I carried on some experiments in a small way at Morenci looking to the development of some "laws" about shapes of mullers, but found nothing better than rods, and since these are easiest to get for the reason above given, the investigation was halted on that ground.

Mine and Mill Plant of the Inspiration Consolidated Copper Co.

BY H. KENYON BURCH, B. S.,* MIAMI, ARIZ.

(Arizona Meeting, September, 1916)

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* Chief Engineer, Inspiration Consolidated Copper Co.

INTRODUCTION

The Inspiration Consolidated Copper Co.'s plant at Miami, Ariz., was designed and built to make possible the profitable working of a low grade, finely disseminated copper deposit containing 100,000,000 tons of ore averaging 1.64 per cent. in copper.

From the beginning it was evident that the plant could not be kept integral but that a break would have to be made somewhere in the flow-sheet, removing at least the concentrator to a site more suitable than any available near the mine. It was finally decided, after considering numerous arrangements, to do the coarse crushing at the mine, to store the crushed ore in a bin from which it could be loaded into railroad cars and to haul it to the concentrator, an excellent site for which was available about 1¾ miles from the mine.

The original intention was to equip a plant to treat 7,500 tons of ore per day, but through the acquisition and proving up of additional ore reserves, the introduction of the Ohio caving system, and the excellent results obtained in the test mill (which made it possible to treat a lower-grade ore than had been thought possible) it was evident that a plant of much greater capacity should be supplied. It was, therefore, decided to treat approximately 15,000 tons of ore per day, the duties of the four main divisions to be as follows:

Division	Operating Time in Hours	Capacity in Tons per Hour	Daily Capacity in Tons	Available Capacity in Tons
Hoisting plant..	15	1,000	15,000	
Crushing plant ...	15	1,000	15,000	
Storage bin.. . .	....	......	.. ...	25,000
Concentrator.. .. .	24	625	15,000	

It was realized from the beginning that with such an enormous tonnage to treat it would be well worth the time and cost to carefully work out a flow sheet. Accordingly, a gravity test mill was erected and placed in operation near the Joe Bush shaft in November, 1910, and its operation was continued until August, 1911. Soon after this, flotation began to attract considerable attention in this country and realizing the possibilities that might arise through its systematic investigation, it was deemed advisable to go into the process in detail. A 50-ton Minerals Separation machine was first erected, which after a few months thoroughly demonstrated that flotation was applicable to the concentration of the Inspiration ores, but in order to better determine its proper place in the flow sheet, a 600-ton test mill was designed and erected in 1913 on the benches of the concentrator site, the grading for which was at this time completed. Numerous flow sheets were experimented with, the final result being the

one now in use in the concentrator. A discussion of certain phases of the results accomplished in the test mill will be given in another section, mention being made of it here simply for preserving the proper sequence.

That the large-scale test-mill method for working out flow sheets for large plants is the only logical method, is evidenced by the fact that in nearly every stage of the treatment either an entirely new machine has been adopted or a new application has been made of a standard machine; the result in each case being either increased efficiency or a more economical arrangement. The first mill was to have a capacity of 7,500 tons per 24 hr. This design covered an area of approximately 350,000 sq. ft., or a little over 8 acres. The mill now in use covers an area of approximately 125,000 sq. ft., or a little less than 3 acres, but has a capacity of 15,000 tons. The recovery in the two types of plants on favorable ores, that is, ores not carrying over 10 points of oxide, may be closely estimated at 70 per cent. for the first, and 85 per cent. for the second.

In order to account for the long period required for the design and construction of the plant, it may be interesting to note that six complete designs for the concentrator were executed, the idea being to keep this work abreast of the developments brought out by the test mill. Design No. 2 was completed and a contract entered into for structural-steel requirements. A portion of the steel had been fabricated when flotation developments pointed to the fact that wet gravity concentration could be greatly improved upon. At this point all work on the steel contract was stopped and that portion of the contract pertaining to the concentrator was cancelled. Although facts relative to flotation continued to develop, in July, 1913, the steel design for the present concentrator building was completed. This building was no sooner erected than very marked changes in grinding machinery began to develop, the ultimate result of which was another altogether new arrangement for the entire mill. As it stands today there are but three single pieces of machinery in the mill building occupying the places originally intended for them, these being the three electric cranes now in use. Considering these changes, it is remarkable that such a good arrangement was found possible.

PLANT SITES

The sites for the plants as finally decided upon permitted of an arrangement entirely adequate to meet the proposed requirements, but the following important features may be noted:

The mine plant site is contiguous to the orebodies but at a safe distance from ground to be caved; it occupies a position as regards elevation well suited to a favorable hoisting arrangement and allows of a downgrade to the concentrator of 0.3 per cent. The shortest line from the shafts to the orebodies approximately bisects them, making possible a

direct underground haulage system; the site occupies a remote corner of the property and does not lie in the general trend of the mineralized zone. The mill site has a slope favorable to the type of mill erected and is of sufficient area to allow of an economical arrangement of the various units of the plant. There is ample storage space for tailings and an excellent reservoir site available for the storage of water.

TYPE OF CONSTRUCTION

As the life of the property was estimated to be 17 years, or longer, depending upon new developments in the treatment of lower-grade ores, it was decided to make the various structures of a semi-permanent char-

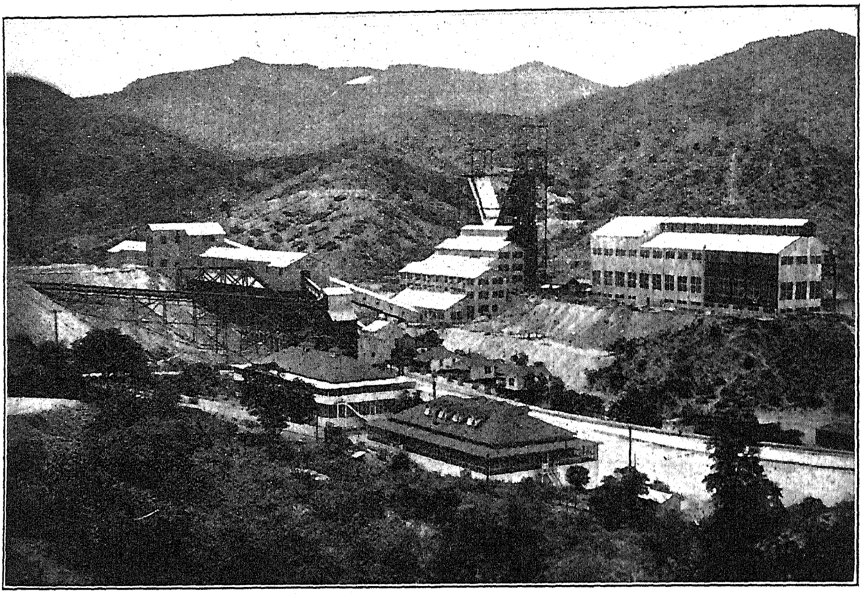


FIG. 1.—MINE PLANT OF INSPIRATION CONSOLIDATED COPPER CO., MIAMI, ARIZ. LOOKING WEST. SHOWING COMPRESSOR HOUSE, MAIN SHAFTS, COARSE-CRUSHING PLANT AND STORAGE BINS.

acter. The buildings are of steel with corrugated steel coverings, except the concentrator which has a four-ply composition roof. All windows as well as skylights are of rubber glass. The floors throughout are of concrete, the retaining walls of reinforced concrete, and machinery foundations of massive concrete with little or no reinforcing. Reinforced concrete was used wherever applicable.

MINE PLANT

Under this general heading will be considered that portion of the plant with its adjuncts which takes the ore from the mine cars, hoists it to

surface, reduces it to a size suitable for mill treatment and places it in storage, ready for transportation to the concentrator.

The relation between the orebodies and the mine plant can be described as follows: Conceive of a chain of orebodies the lateral extensions of which form a parabola with its vertex pointing to the north and having a length of approximately 9,000 ft. Place the main haulage ways on either side of the axis, 60 ft. apart, and open them to surface through two three-compartment shafts, 102 ft. centers, and about 750 ft. from the nearest approach of the orebodies. Group the mine plant symmetrically about the axis between the shafts and the orebodies and a general idea of their relation is obtained.

Dual Arrangement

As a result of the preliminary studies, the dual arrangement of the mine plant was evolved, the primary consideration being the necessity of handling 1,000 tons of ore per hour, which in itself precluded the use of a single shaft. Two shafts would also insure continuity of service. It was then considered advisable to follow this idea a step further and make the whole mine plant duplicate in its arrangement, which would give reasonable assurance against total shutdowns and also permit of a better load factor.

Underground Haulage

Underground haulage, for transporting crude ore from the stope chutes to the loading stations at the main shafts, is confined to two levels, the 600 or main haulage level, and the 400 level. The main haulage level is, in turn, made up of two distinct systems, each serving its main shaft. All haulage is to be controlled by a block signal system.

Two types of motive power for underground haulage were considered —electric and compressed air. On account of less danger to life due to the elimination of bare conductors, the compressed-air locomotive was chosen rather than the electric. The difference in the efficiency of the two systems is probably a small percentage of the total cost. "Safety First" was therefore the deciding factor.

The locomotives are of the two-stage, four-wheeled type, and have a weight on the drivers of 10 tons. The initial cylinder air pressure is 250 lb., and the charging pressure 800 lb. per square inch. The haulage capacity per charge was estimated at 50 ton-miles. The locomotives haul 25 cars, each of 5 tons capacity, on a 30-in. gage track over a 0.4 per cent. grade in favor of the load, and will negotiate a curve of 38 ft. radius.

All ore tapped from the stope chutes is broken to pass a 12-in. grizzly so that no further attention has to be given to it after it is once in the cars. At the shafts the cars, weighing 14,000 lb. each loaded, are dumped, five at a time, by motor-operated tipples, one for each shaft on the 600

or main haulage level, with a third on the 400 level. All hoisting is done from the main haulage level.

Tipples

The tipples have an overall length of 56 ft., and make a complete revolution in 15 sec. The driving shaft is connected through suitable gearing to a 35-hp. motor which runs continuously in one direction, the starting and stopping of the tipple being accomplished by means of a friction clutch located on the intermediate shaft. Each tipple is provided with an automatic stop which brings it into the proper position for running on and off the cars, this stop being released by means of a foot lever on the operator's platform. The two main tipples are operated from a centrally located platform.

Underground Pockets

The tipples on the 600 level dump directly into two main underground pockets (Fig. 2), which serve as storage for the four automatic measuring and loading devices that are placed beneath them. The capacity of each pocket is 1,600 tons, or 800 tons for each loader, which, with full pockets, will run the crushing plant about 3 hr. They are of reinforced-concrete construction throughout and are lined with 2-in. planks. The upper pocket has a capacity of 500 tons and is connected to the lower pockets by an inclined chute.

Automatic Measuring and Loading Devices

To carry out the automatic feature of the hoisting equipment, it was desirable to make the loading of the skips automatic. There being nothing on the market which could be used for this purpose, it was necessary to work up complete designs in accordance with original ideas.

In a few words the device (illustrated in Figs. 3*a* and 3*b*) can be described as follows:

Ore from one of the 1,600-ton underground pockets rests on a roll feeder which is actuated by a pawl and ratchet wheel, and is fed into a 12-ton measuring hopper. This hopper is suspended between the legs of a U-shaped scale beam with fulcrums at the two ends. Through a link at the middle of the yoke the proportional weight of the hopper and ore is transferred to the main scale beam which carries a counterweight of 1,370 lb. Motion obtained from the counterweight scale beam when the hopper becomes filled is utilized to operate a system of levers and lift the pawl from the ratchet wheel, thus stopping the roll feeders. The hopper is now ready to be emptied.

As the returning empty skip settles on the chairs, it strikes an arm

projecting out into the shaft. This arm is connected by a system of links to the shaft on which the yoke of the gate forming the bottom of the hopper is hinged. The motion imparted to the system by the skip trips the gate yoke, the weight of the ore in the hopper causes the gate to drop

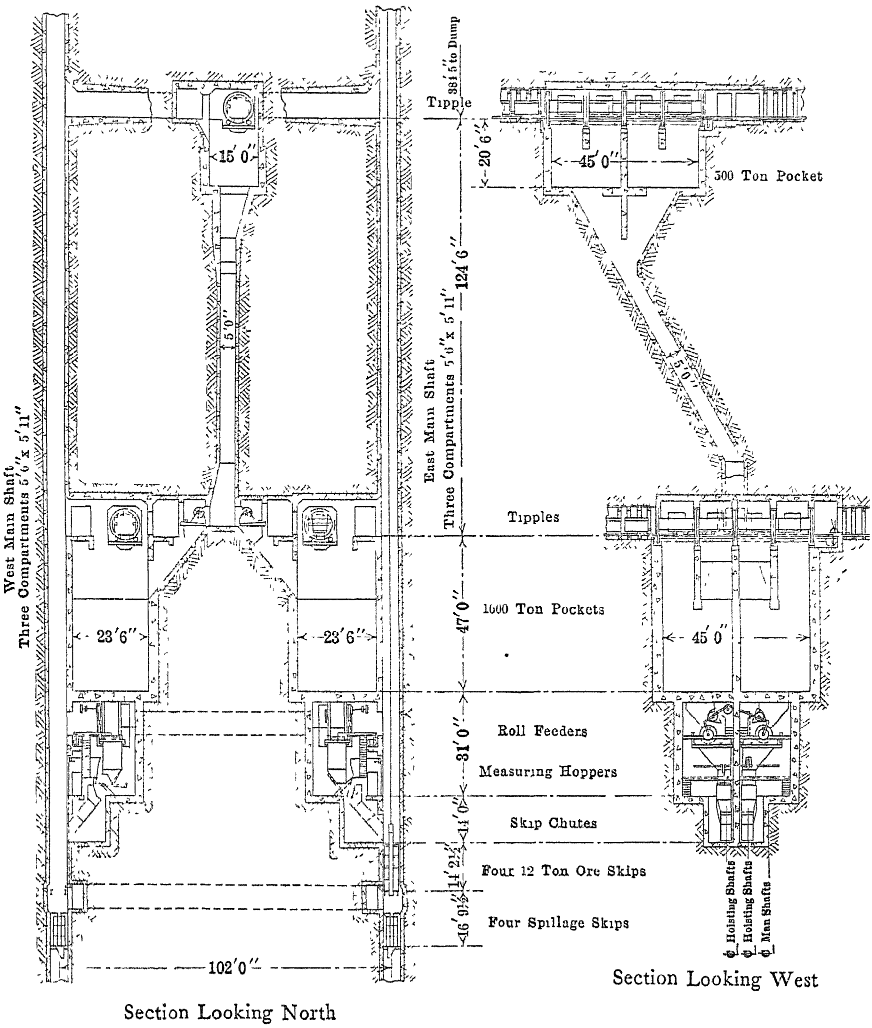
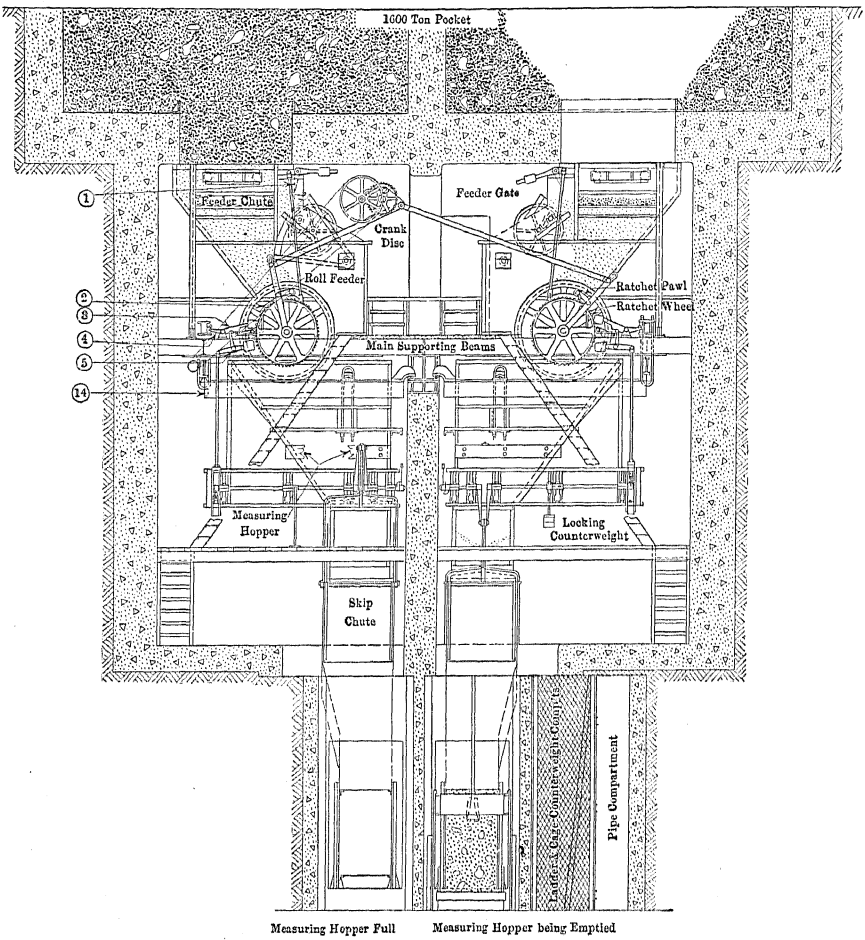


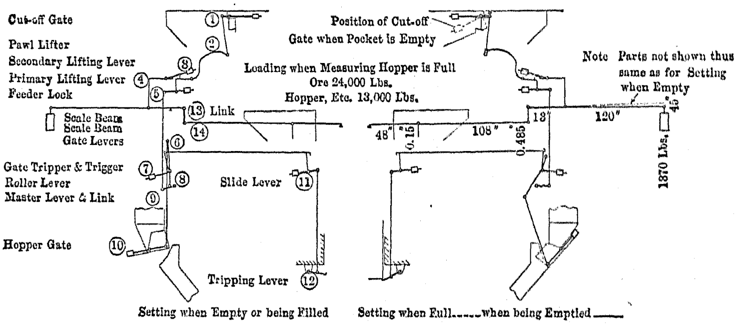
FIG. 2.—ARRANGEMENT OF UNDERGROUND POCKETS.

into the skip chute, and the hopper is emptied into the skip. As soon as it is empty, a counterweight on the gate causes it to close, all motions are reversed and the measuring hopper is again filled.

A cut-off gate prevents the roll feeder from becoming bare in case the pocket should be emptied. A 5-ft. layer of ore thus protects the feeder

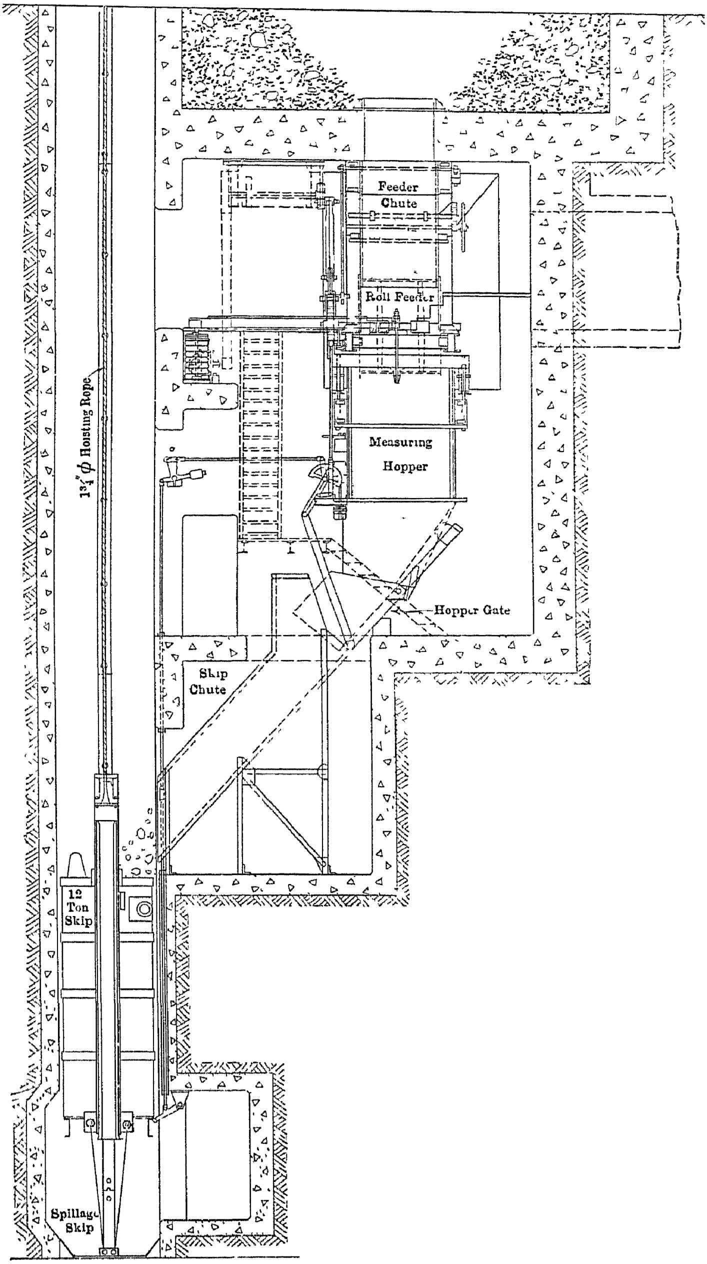


Sectional Front Elevation



Line Diagram showing Principle of Operation

FIG. 3a.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.



Sectional Side Elevation
Measuring Hopper being Emptied Feed Cut-off

FIG. 3b.—AUTOMATIC ORE MEASURING AND LOADING DEVICES.

and does away with excessive impacts due to the dumping of the ore from the tipple 50 ft. above. A 5-hp. motor furnishes the required power for the roll feeders of each pair of loaders.

Shafts

Taken individually the shafts do not deviate materially from usual designs, being of reinforced concrete throughout. They are of the same size, each being made up of three compartments 5 ft. 6 in. by 5 ft. 11 in., the two south compartments of either shaft being for the ore skips. The north compartment in the West Main serves the man elevator and the corresponding compartment in the East Main is taken up by pipe lines, ladderways and the elevator counterweight. All platforms, supports, and ladders in this compartment are of steel. The conduits carrying the power and lighting circuits underground are in the end walls, being brought to junction boxes every 100 ft.

Ore and Spillage Skips

The ore skips are built according to special designs. They have a nominal capacity of 12 tons and weigh 17,000 lb. each. The overall length is 21 ft., the skip proper being 14 ft. deep. The body is of $\frac{3}{8}$-in. steel plate with liners to take up concentrated wear.

In order to keep the sumps at the bottom of the shafts clean and to facilitate the handling of ore which does find its way below the skips, spillage skips have been installed (Fig. 2). These rest on chairs 17 ft. below the ore-skip chairs and by means of hinged aprons completely close off the shaft. When one is filled it is attached to the bottom of the ore skip and hoisted to the top of the underground pocket, where by opening a hopper bottom it is discharged. They have a capacity of 9 tons each.

Automatic Hoists

At the time it was decided to sink two shafts the conditions seemed to favor the use of two entirely independent hoists, located symmetrically on either side of the main headframes. It was next proposed to put the two hoists under the same roof but to operate them independently of each other. Then on account of the advent of several new conditions it was deemed advisable to make the operation of the hoists automatic.

While the hoists are entirely automatic in their operation, either can be manually operated independently of the other if so desired. No attempt will be made to cover the automatic features of the installation as these are presented in detail in another paper.[1] Suffice it to say here

[1] H. KENYON BURCH and M. A. WHITING: Automatic Operation of Mine Hoists Exemplified by the New Electric Hoists for the Inspiration Consolidated Copper Co., this volume, p. 10.

that the problem as presented has been solved in a most efficient manner, sufficient proof being the entire satisfaction that the equipment has thus far given.

Man Elevator

When the question of handling men came up, several possibilities as regards point of distribution presented themselves. They could be handled through an existing tunnel, hoisting or lowering to the working levels through an inclined shaft, or entirely through an inclined shaft, or through one of the main shafts. Whichever scheme proved the most feasible, the idea was to have the men do as little walking as possible, ample provision to be made for transferring them from change house to working places. It was finally decided that distribution from the main shafts was the most logical and the shafts were equipped accordingly.

The next consideration was the type of hoists to be used. Standard mine practice favored a separate hoist, manually operated and working against a counterweight. This type made necessary the services of two attendants, one on the cage and one at the hoist, but in working out probable operating costs the idea was conceived of eliminating the hoist attendant by an application of the elevator principle. Studies along this line were at once begun, but no great amount of enthusiasm among the manufacturers could be aroused on account of the unusual conditions. Equipment was wanted to handle a counterweighted, double-deck cage, weighing 7,100 lb. and loads of 7,500 lb., the maximum speed of the cage to be 800 ft. per minute. The fact that the hoist and counterweight would have to be removed from the elevator shaft also presented a new feature in the design. It may be said, however, that placing the hoist in the headframes directly above the shaft, as is the usual practice in office building design, was at one time considered.

The equipment as installed is operated entirely from the cage, the hoist, motor-generator set, and control apparatus being in the hoist house about 220 ft. away. The motor driving the direct-current generator on the motor-generator set has a rating of 190 hp., the generator developing 130 kw. The hoist is driven by a 158-hp. direct-current motor. It is provided with a safety brake, so arranged that when the hoist is stopped the brake is automatically applied to hold the cage. This brake is actuated by spring pressure, is constantly in service except when electrically released during normal operation of the hoist and is, therefore, instantly applied in case the current supply is interrupted from any cause. Additional safety devices are provided which take care of overwind, overload on motor, slack cable, etc. Two oil-cushioned buffers are provided at the bottom of the shaft, being so designed as to bring the loaded cage to a gradual stop in case the cage from any cause, while running at normal speed, should not stop at the lower terminal. The cage is equipped with

a telephone which keeps the attendant in touch with the hoist house, also with an annunciator system covering the several levels.

Each level and the collar are provided with two-story stations that make it possible to load both decks without shifting. The cage accommodates 36 to 40 men besides the attendant. The surface station, which is in the headframe, is connected with the change house by a covered runway. From the underground stations the men are transferred to the various working places in cars accommodating 12 men each and drawn by the haulage locomotives.

The flow sheet in Fig. 7 should be referred to in connection with the following description of the surface plant of the Inspiration company.

Crushing Plant

The loaded ore skips assume a dumping position at a point about midway between the shaft collars and the center of the sheaves, leaving about 25 ft. for possible overwinding. The sheaves are 125 ft. above the shaft collars. They are 12 ft. in diameter and are pressed on 8½-in. shafts; 8-ft. sheaves on 6-in. shafts are used for the man-elevator ropes. The dumping tracks are of usual design.

Crude-ore Bin

By means of chutes leading away from one compartment of each shaft, an even distribution of the ore is effected throughout the length of the 2,000-ton crude-ore bin. This bin is of rather unusual design and very nearly self-cleaning. The bottom slopes down 45° each way from a hip center, toward the vertical back and front, a section resembling an inverted capital M. Gates are placed under the bin on the slope toward the back as well as on the front of the bin. This arrangement of bottom and gates not only allows the bin to be emptied but the ore is prevented from packing or hanging up by drawing off alternately from the rear and front gates. The gates are 48 in. wide and are operated by rack and pinion from runways.

For each of the four units of the crushing plant, two gates, one under the bin and one in front, discharge on an apron feeder that travels across and under the bin, delivering the ore upon a 3-in. bar grizzly feeding a No. 8 gyratory crusher set to crush to 4-in. cubes (Fig. 4).

The apron feeders have a width of 48 in., a length of 25 ft. 6 in. between centers and travel at 7 ft. per minute; 20-in. belt conveyors traveling at 7 ft. per minute and driven from the feeder sprockets are placed beneath the feeders to take care of drippings. These discharge into chutes that join the undersize from the grizzlies. Wood scraps are removed from the ore at the discharge end of the apron feeders and also

from the inclined conveyors leading to the disk crushers. These scraps are carried away by two 20-in. belt conveyors traveling at 75 ft. per minute, running across the building and dumping outside.

One of the four similar crushing units will be described in the following paragraph. Fig. 4 shows a cross-section of the crushing plant.

Description of Crushing Unit

The discharge from the gyratory and the undersize from the grizzly are conveyed by a 30-in. inclined belt conveyor to two 48-in. disk crushers,

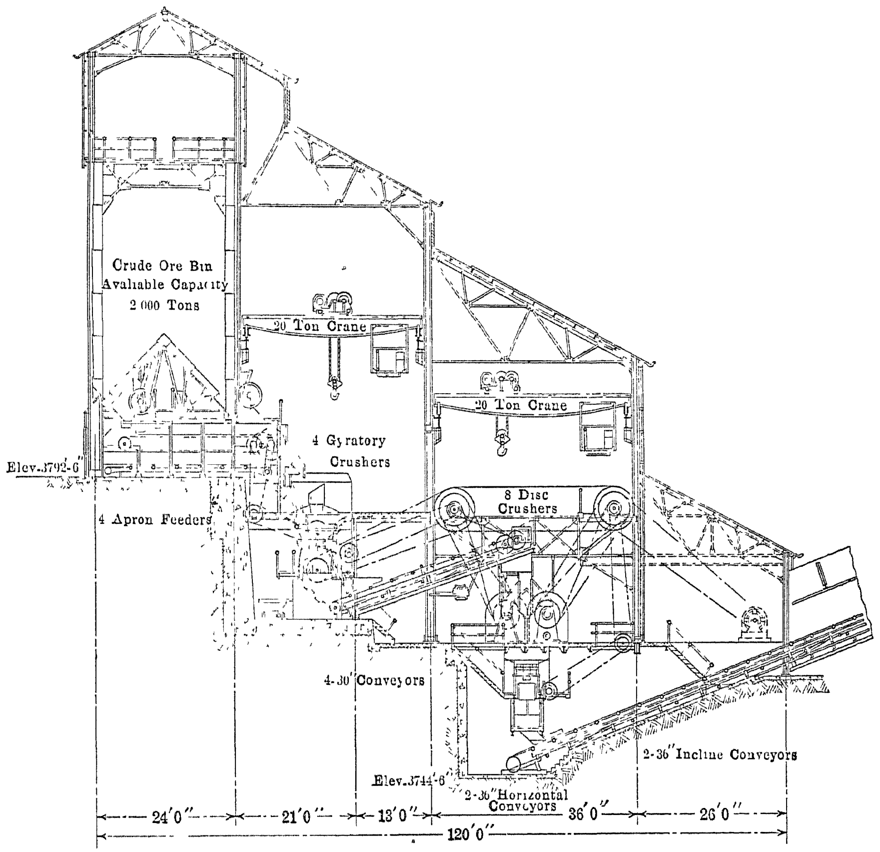


FIG. 4—SECTION OF COARSE-CRUSHING PLANT.

the ore first passing over a 1½-in. bar grizzly. This conveyor has a magnetic head pulley for removing tramp steel. The 48-in. disk crushers are set with a maximum opening of 2 in., their discharge and the undersize from the 1½-in. grizzlies joining on two 36-in. horizontal cross belts which in turn discharge at the center of the building on the two main

inclined belts, also 36 in. in width. These belts are 300 ft. long, travel at 350 ft. per minute and are equipped with hand-adjusted tail-pulley take-ups and also with weighted tension carriages. They discharge on four 24-in. horizontal conveyors over the storage bin, each of which has an automatic tripper which distributes the ore uniformly throughout the bin. Weightometers are placed on the two main inclined belts, thus giving a record of the ore handled.

Storage Bin

The purpose of the storage bin in connection with a mine plant is identical with that of a flywheel on an engine: "To give out and absorb energy when variation in the load occurs suddenly." It does away with the necessity of maintaining a nice balance between mine and mill and allows either, for a short time, to cease running altogether, or to run at a lessened capacity. Where the mill is removed from the mine plant, it also furnishes an economical means of transferring the ore into railroad cars. Besides this it allows of a hoisting and crushing period considerably less than full-time running, a very important consideration, since there are always repairs to be made in shafts, to loaders, tipples, hoists, and crushing machinery.

The Inspiration storage bin is double tracked, has an available capacity of 25,000 tons and will load a train of 14 cars without switching. The base is of reinforced concrete, and is provided with four expansion joints. The bin proper and conveyor housing are of steel. The main dimensions of the bin are as follows: Overall length, 465 ft.; width, 40 ft.; depth, 40 ft. The bin is divided into three compartments, the two end compartments being for the storage of special ores if any such ever be encountered. They are comparatively small, accommodating one car on each track and having an available capacity of 1,700 tons each.

The distributing belts are driven by two 50-hp. motors located at either end of the bin. The ore is drawn off through hand-operated gates of the swinging cut-off type, there being three gate openings for each car.

Sampling Plant

The heads sampler is placed at the point where the main inclined belts discharge on the horizontal cross belts (Fig. 7). This sampler consists of a rectangular cast-iron bucket, 9 by 30 in., on the end of a revolving arm 10 ft. in length. The bucket runs on a circular track 12 ft. 4 in. in diameter and is driven through a wormwheel mounted on a vertical shaft, making a revolution every 40 sec. The bucket has a swinging cut-off type of gate for a bottom. In its travel it cuts the two main ore streams, taking out $\frac{1}{120}$, and at a point 180° from the ore stream strikes a tripper

which causes the gate to open and the sample to be discharged into a 10-ton hopper. From the hopper the sample is fed by a roll feeder and started on its way through the sampling plant, which is located against the center of the storage bin opposite the main conveyors. Here the sample is first passed through a 24-in. disk crusher. A 1/15 cut is then made with a Snyder sampler and this cut is delivered to a set of 27 by 14-in. rolls. The roll product is cut by a second Snyder sampler, this 1/15 cut going to a mixing drum that equalizes the flow. The resulting stream is delivered to a duplex Vezin sampler making two 1/10 cuts. These samples are duplicates and weigh 110 lb. each, representing a day's run. The rejects from the three samplers go to an elevator that discharges into the storage bin.

Motors and Other Equipment

The four units of the crushing plant are driven by four 200-hp. motors located in two dustproof rooms, symmetrically arranged on either side of the plant. Except for some minor reductions, belt transmission is used throughout.

Two 20-ton electric cranes are installed in the plant, one to serve the gyratories, the other the disk crushers, motors, etc. These are in turn connected to the yard track by transfer cars and a 20-ton electric trolley hoist, running at right angles outside of the building. Each 200-hp. motor is served by a 6-ton crawl so arranged as to allow of crane handling after leaving the motor room. A repair car and track near the gyratories permits of easy handling of their driving mechanisms, or bottom plates.

Compressor Equipment

Compressed air is used for two distinct purposes: For underground haulage, and for mine drills. For the first, 1,000-lb. air is used and for the second, 100-lb. The haulage system is served by two compressors, each having a capacity of 1,125 cu. ft. of free air per minute at 107 r.p.m. These are of the duplex, four-stage, power-driven type, the cylinders being cross-compounded, water-jacketed, and having intercoolers between the stages. These compressors have an automatic regulating arrangement whereby the 1,000-lb. discharge line is cut out when a predetermined high-pressure point has been reached, and the four cylinders converted into a double-compound compressor, discharging directly into the 100-lb. main. Each compressor is driven by a 430-hp. self-starting, synchronous motor.

The low-pressure equipment consists of two 100-lb. compressors having capacities of 3,000 and 7,270 cu. ft. of free air per minute. Both compressors are of the parallel, two-stage type, all cylinders being water-jacketed with intercoolers between the two stages. The 7,270-ft. machine

is equipped throughout with Hoerbiger-Roegler plate valves, which on account of the very short movement and exceedingly light construction permit of a high compression efficiency. The cylinders of this machine are 46-in. and 28 by 36-in. stroke. It is driven by a self-starting synchronous motor of 1,150 hp. The 3,000-ft. machine is driven by a 500-hp. self-starting, synchronous motor at 107 r.p.m.

Two 100-kw., 125-volt, motor-generator sets arranged to operate in parallel serve as exciters to the compressor motors. Cooling water for the jackets and intercoolers is obtained from a circulating system consisting of a duplicate installation of 750-gal. triplex pumps and a spray cooling pond.

The compressors and hoisting equipment are housed in a spacious and well-lighted building which has a 25-ton electric crane running its entire length connecting at one end with a well into which railroad cars can be run and unloaded. A 22-panel, remote-control switchboard, together with the low-tension (2,200-volt) bus equipment, occupy a room along one side of the building, the high-tension transformer station being outside near one end of the building. Only two attendants per shift are required, one being an oiler. Without the automatic features for hoisting ore and the elevator for handling men, at least four would be required.

CONCENTRATOR

The considerations which led to the type of plant finally decided upon will first be given; this will be followed by a description of the plant. The concentrator includes the concentrates filter plant. Fig. 5 shows the concentrator with its adjuncts.

600-Ton Test Mill

The underlying principle that pervaded the work in this mill was to carry on all experiments with the idea constantly in mind that the particular method or machine under test might eventually be installed in the new mill.

The test mill was put up primarily to try out flotation on a large scale in order to determine its proper application to Inspiration ores. Since the flotation process, as applied to the concentration of copper ores, was an innovation, it was logical to suppose that in its last analysis the flow sheet might differ altogether from wet concentration methods. This did not prove to be the case, but the assumption was made and it was for this reason that the test mill was made so flexible.

The test mill was started in February, 1914. The concentrates sold have paid the expenses of erection and operation, a remarkable record for a test plant. It has treated from 600 to 1,000 tons per day, the amount depending upon the types of machines being tested.

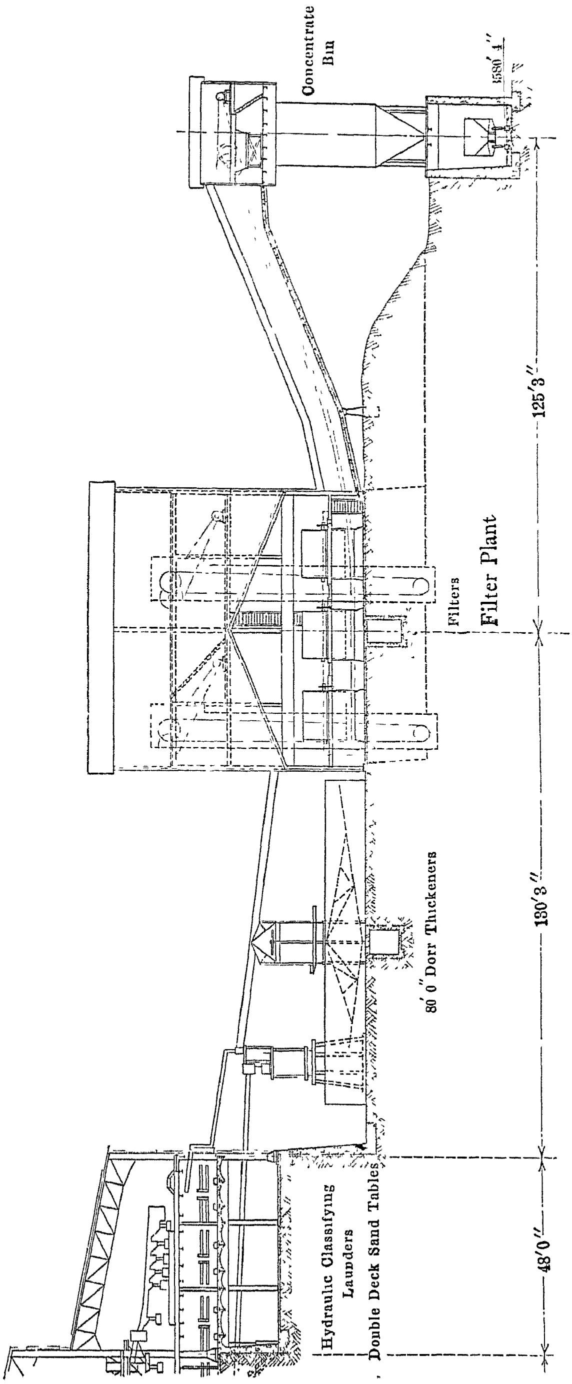


FIG. 5.—SECTION OF CONCENTRATOR AND FILTER PLANT.

The building is of timber, covered with corrugated steel. This type of building was chosen because it would facilitate changes or additions, was the cheapest and quickest form of construction, and would still have considerable value when dismantled. All machinery was placed on concrete foundations.

The ore for the test mill was taken from a development shaft and reduced to 3-in. cubes in a No. 8 gyratory crusher located nearby, the crushed ore being loaded into cars which emptied into a 180-ton bin at the head of the test mill. The following description gives the way in which the mill was first run, the various additions and changes being noted in sequence.

Two 30-in. apron feeders delivered ore from the bin to an inclined 20-in. belt conveyor, which was equipped with a recording weighing machine and a magnetic head pulley. This conveyor delivered the ore to a 36-in. Symons horizontal disk crusher and a Symons 48-in. fine reduction disk, or vertical disk crusher. These two machines delivered to a second 20-in. inclined conveyor. If so desired the two crushers could be bypassed, the delivery then being made direct to the second conveyor. This permitted the crushers below the second conveyor to be tried either as primary or secondary machines. The second conveyor delivered the ore to another 48-in. fine reduction disk and a Symons 48-in. roller mill. Water could be added either above or below these crushers. The ore then went to four Hardinge mills, the feed to which was regulated within suitable limits by a mechanical distributor. Three of these mills were 8 ft. in diameter with barrels 36 in., 44 in. and 72 in. long, all being direct-driven by induction motors. The fourth mill was 10 ft. in diameter, had a barrel length of 28 in. and was belt-driven. All of these machines were pebble mills with either silex or El Oro linings.

A drag classifier followed each mill, the whole floor being so arranged that each mill could be placed in a closed circuit with its drag or two mills and their drags run in tandem, that is, the first mill discharged into a drag, the sand from which was delivered into the second mill which in turn discharged into the second drag. The sand from the second drag was returned to join the sand from the first drag to form the feed for the second mill. The overflow from the drag was delivered either to eight sand tables by two distributors, or sent to other distributors which fed eight slime tables at the lower end of the mill.

Between the sand and the slime tables were placed the flotation machines which could be fed directly from the drag overflow or by any table product. The original installation consisted of two Minerals Separation eight-compartment flotation machines, one of 50-ton and the other of 600-ton capacity. By means of elevators any flotation product could be retreated on the slime tables or in the small flotation machine.

The concentrates from the tables and flotation machines went to a

filter plant containing an Oliver and a Trent filter. The dried concentrates were loaded into 1-ton cars, weighed and dumped into a railroad bin from which they were taken to the smelter.

Mechanical samplers were installed at important points and hand samples taken where necessary.

The crushing and grinding experiments were not only extensive but proved most interesting, the results being quite unexpected. The 36-in. disk crusher represented the 48-in. disk crushers now installed in the crushing plant following the gyratories.

As secondary or intermediate crushers, that is, machines between the 36-in. Symons disk and the Hardinge mills, the following were tried out: The Symons 48-in. fine reduction disk; the Symons 48-in. roller mill; the Symons 56 by 48-in. ring mill; the Bradley 66-in. centrifugal roller mill; the Overstrom centrifugal crusher; the Allis-Chalmers No. 4 hammer mill; and the Marcy ball mill.

The final results indicated that the Symons fine reduction disk, with a little redesigning, would be an efficient machine for the limited field of crushing from 4-in. cubes to 4-mesh. It was found that this could be done with a single pass, no oversize resulting. The machine would not work well wet and it was found necessary to remove all fines from the feed to the machine, otherwise a packing would occur. Consumption of steel wearing parts appeared low, as also did power consumption, although no definite results as to these two points were obtained.

The Marcy mill also proved to be a most excellent intermediate crusher, handling as high as 800 tons of 3-in. feed and under to pass an 8-mesh screen without return of oversize.

To compete with the four Hardinge mills a 6 by 20-ft. Chalmers & Williams tube mill and an 8 by 5-ft. ball mill were installed. The latter was a Marcy mill using 5-in. diameter steel balls. It was placed in a closed circuit, as were the pebble mills, and gave good results. It was next tried on a feed from the 36-in. disk crusher and then on a feed direct from the gyratory. These experiments definitely demonstrated that only three reductions were necessary: A gyratory, followed by a disk crusher and ball mills in a closed circuit. This is the arrangement installed in the present plant.

Several different classifiers were tried in the ball-mill circuit, a specially designed duplex Dorr classifier being adopted.

The flotation machines and the tests conducted on them are described in another paper,[2] but it is interesting to note here that the following machines were tried out: Minerals Separation, Hoover and Hebbard types; the Callow flotation cells; the Cole-Bergman; and the Inspiration machine.

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[2] Rudolf Gahl: History of the Flotation Process at Inspiration, this volume, p. 576.

The Inspiration machine was devised during the operation of the test mill.

The concentrator is equipped as follows: 4 units with the Callow machines; 13 units with the Inspiration machines; and 1 unit with the Minerals Separation machines.

The drag classifier as originally tried out proved to be too small for the tonnage required, the drag now in the concentrator having been developed from the small one.

The tables tested were the Deister Machine Co.'s double-deck slimers, double-deck sand tables, and four-deck slimer; the Deister Concentrating Co.'s single-deck slimer, double-deck sand table and No. 4 sand machine; and the Wilfley No. 6 table with decks arranged for both sand and slime. The Deister Machine Co.'s double-deck sand table was adopted, 198 being installed.

This sketch of the test mill does not bring out the many small things experimented with, such as shaking screens, Caldecott cones, samplers, feeders, slopes of launders and conveyors, or the necessary efforts to work out improvements on the new machines.

The 15,000-Ton Concentrator

The ore comes from the mine plant in 14-car trains, and is dumped into the concentrator bins, which have a capacity of 12,000 tons, or 670 tons per unit (Figs. 5 and 6). These bins are of the suspension type, are double tracked, and extend the full length of the mill, 300 ft.

The mill is made up of 18 units, similar in equipment except for the differences in the flotation department. Eighteen 30-in. apron feeders, located on the center line of the bin, feed the ore to 20-in. inclined conveyor belts traveling at 150 ft. per minute. Weightometers are installed on these belts, which record the amount of ore fed to each unit. The conveyors discharge into hoppers where a split is made, half of the feed going to the north ball mills and half to the south mills, the transfer being made through launders with cast-iron liners and having a slope of 45° (Fig. 7).

Grinding Section

Floor Arrangement.—One of the novel features relative to the layout of this plant is the arrangement of the grinding floor. With due regard for the individual capabilities of the machines used, it is nevertheless a fact that the arrangement has been important in bringing about the excellent results obtained. The following tabulation will serve to show that the method, besides being efficient in the work it is doing, is conservative of floor space as well:

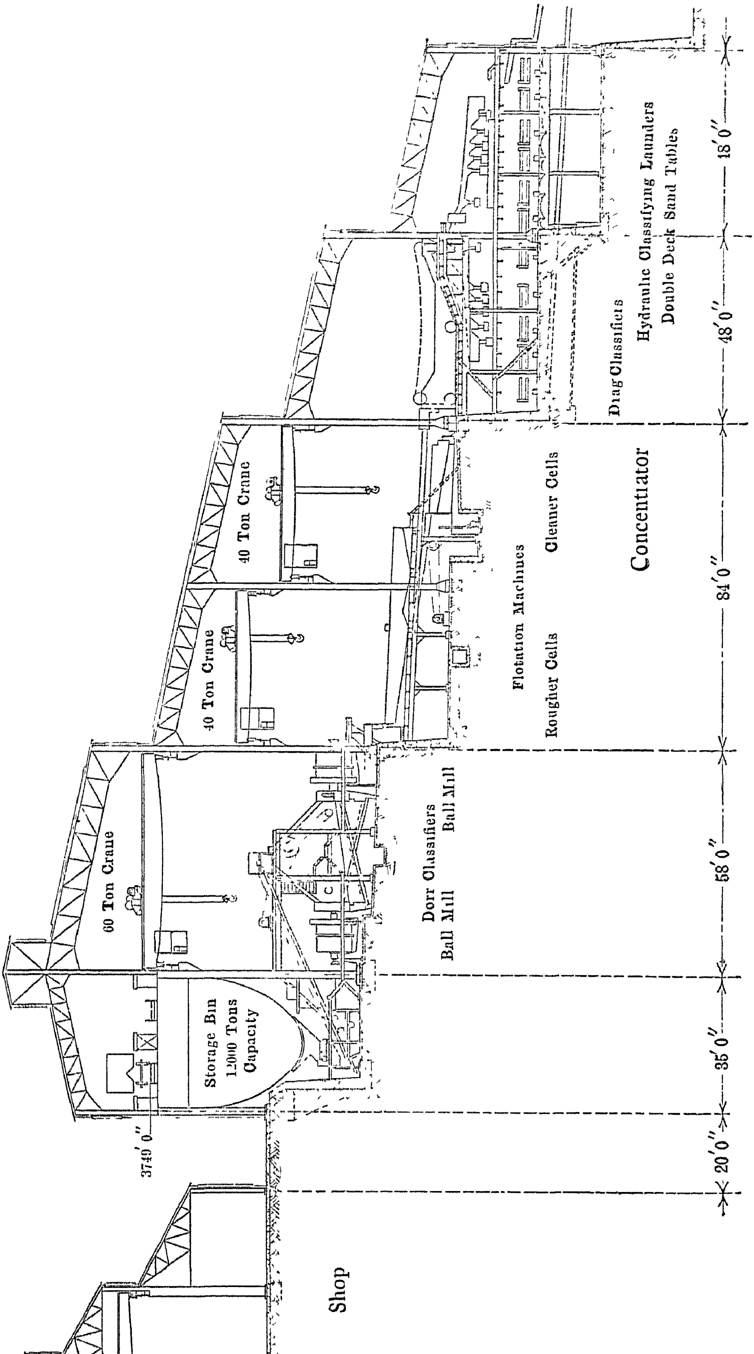

FIG. 6.—SECTION OF CONCENTRATOR.

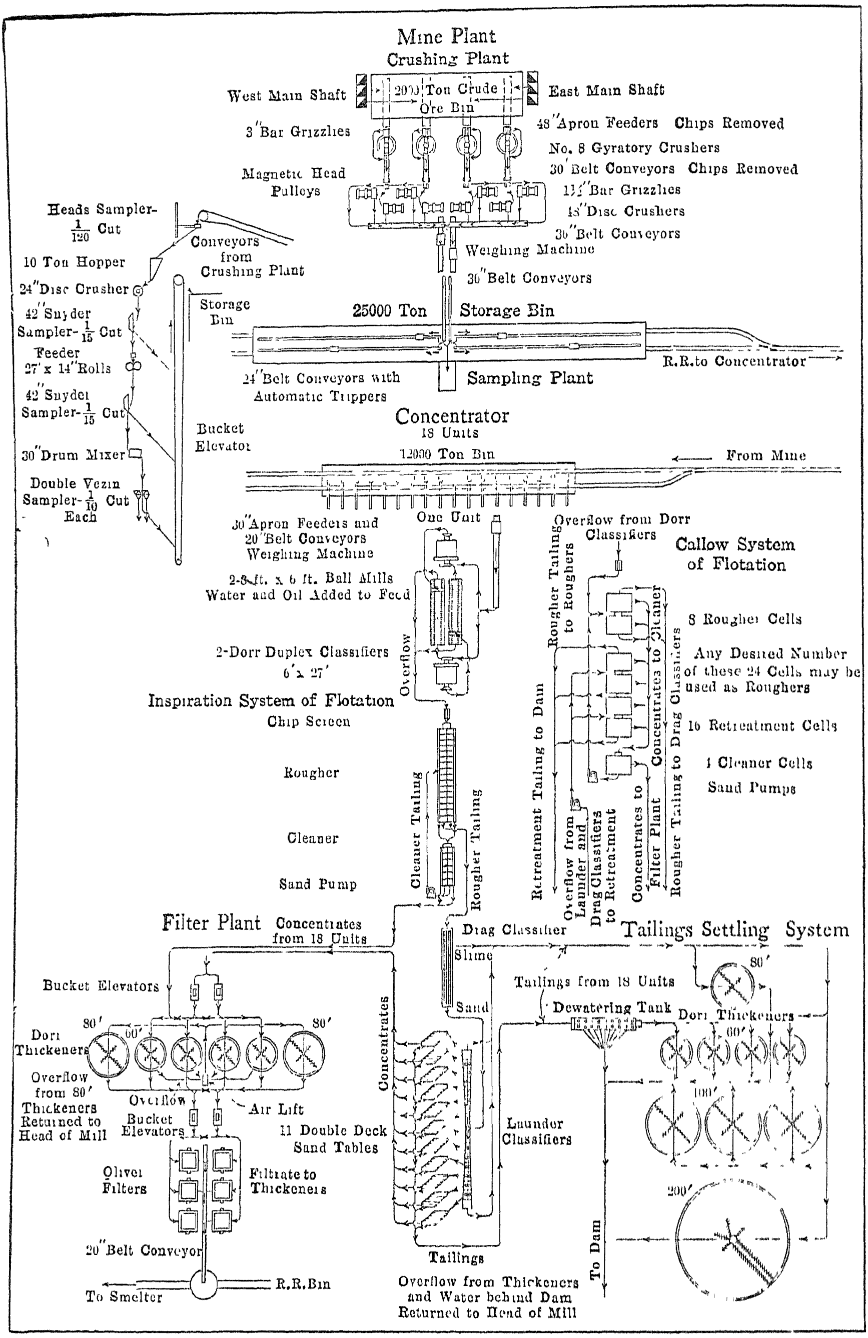

FIG 7.—FLOW SHEET OF INSPIRATION PLANT.

Capacity of installation, 15,000 tons per day from 2-in. disk crusher opening to 1½ per cent. on 48-mesh.

Area of grinding floor, exclusive of motor platforms, 300 by 66 ft., or 19,800 sq. ft.

Floor space per ton capacity, 1.32 sq. ft.

The scheme can be briefly described as a parallel-series arrangement; two circuits—each with a Marcy ball mill and a Dorr classifier in series—are arranged in parallel, the oversize from each classifier being returned to the mill opposite for regrinding. The mills are symmetrically arranged in two rows extending the length of the mill with their feed ends facing each other, just enough room being left between them to accommodate the Dorr classifiers (Figs. 10 and 11).

The Dorr classifiers act as elevators and, since all return is handled by them, the whole department is confined to a single level. There is a main runway between the mills above the classifiers, from which branch runways are taken off to lead to the motors, between mills, etc. This arrangement gives the attendant an excellent opportunity to watch his machines and to reach them when necessary. The conveyors bringing the feed to the mills are terminated in hoppers on a platform high enough above the mills to allow of a 45° slope for the launders leading to the feed boxes. The motors for driving the conveyors, classifiers, etc., together with their control apparatus, are located on this platform, as are also the control panels for the ball-mill motors. The switchboard attendant thus has a commanding view of the whole grinding floor. The floor is served by a 60-ton electric traveling crane which transfers the mills to and from the repair floors located at either end of the building. An inclined skipway along one side of the building serves the whole concentrator, the cranes on each floor making direct connection with it.

Marcy Ball Mills.—The feed for each ball mill consists of ore from the bin, together with the sand from the discharge of the mill opposite, the final product of the system being the overflow of the classifiers. The feed to the ball mills has a consistency of about 1 to 2, and the overflow from the classifiers about 3 to 1, the latter being maintained by an automatic device which adds water as required in the discharge boxes of the ball mills.

The ball mills are 8 ft. in diameter and 6 ft. long. They are direct-driven through herring-bone gears by 225-hp. induction motors. Our experiments with the ball mill in the test plant led us to the conclusion that we could crush 400 tons of the coarse-crushing plant feed with each mill in 24 hr. to 2 per cent. on 48-mesh screen and 60 per cent. minus 200. Now that the Inspiration mill is running with all sections, it has been shown that our test figures were correct. Not only have we been able to crush 400 tons per mill, but with 18 sections running the average daily dry tonnage for the month of May was 15,358. We have a test

section (of two mills) that showed for a short period a tonnage rate of 1,000 tons per 24 hr. While this is by no means the average, yet it indicates what may be done in the future.

Performance of Crushing Machines.—In view of the character of the ore and the fact that much of it is tough and more difficult to crush than the ordinary porphyry ores, we consider the power consumption for our crushing plants very satisfactory. The power consumption at the coarse-crushing plant for the month of May, taking mine-run ore, crushing it in gyratories and disk crushers, and conveying to the railroad bins, was 0.329 kw.-hr. per ton. At the mill, the average power consumed per ton during the month of May for the ball mills, including power for feeders, conveyors and classifiers, was 10.4 kw.-hr. Therefore, the total power consumed per ton for the month of May on ore taken from the mine bins, crushed and delivered to the flotation machines was 10.729. We have had one section of the ball mills in operation for a period of 10 days that has given a power consumption of 8.53 kw.-hr. per ton. If we add to this figure the power used at the coarse-crushing plant and of the conveyors, we would have 8.88 kw.-hr. per ton of mine-run feed delivered to the flotation machines. The above does not include the power required for transporting the ore by steam road from the mine to the mill.

The crushing and delivery of the ore for concentration is a matter of great importance, and one that absorbs a large portion of the operating costs in the treatment of ores. Too much attention cannot be given it. We are now conducting many experiments in connection with our crushing problem, including different types of steel for wearing parts and modifications that influence our tonnage and cost of production. We have equipped ball mills without the grates and the results of the mills so equipped have shown us that ball mills equipped with grates have advantages in increased tonnage and efficiency. Our experiments with the different shapes of pebble mills, our subsequent experiments with the ball mills, and finally our monthly operation, lead us to the conclusion that a good choice was made in the selection of our present grinding machinery.

Flotation Section

The overflow from the Dorr classifiers goes to the flotation machines, of which there are three types now in operation. Generally speaking, the flotation machines make a clean concentrate which goes to the filter plant, a middling which goes to drag classifiers and a tailing which goes to waste. As the subject of flotation is to be taken up in detail in another paper, just enough will be given here to bring out the part the process plays in the general scheme. The flow sheets for the two processes differing somewhat, each will be considered separately.

Flow Sheet with Callow Flotation Cells.—A unit installation of these machines consists of a total of 28 cells which for sake of compactness are grouped into sets of four cells each, individual cells being 3 ft. 3½ in. by 10 ft. 2 in. Of this total, eight are roughers, 16 are for retreatment and four are cleaners, the arrangement of the group being such that all feeds and products except two are handled by gravity. These two are elevated by sand pumps. The Callow cells, as installed in this plant, make two products, a concentrate and a tailing.

The overflow from the ball-mill classifiers first undergoes a roughing treatment in the eight roughers. Here the two products are a concentrate which goes to the four cleaners and a tailing which goes to a large, specially designed drag classifier, a detailed description of which will be given later. The concentrate from the cleaners is a finished product and goes to the filter plant. The tailing from these cells is pumped back into the rougher cells for a second passage through the system. The tailing from the rougher cells is classified in the drag into two products, a sand and a slime overflow, the sand being still further prepared for table concentration in hydraulic classifiers. The tables make two products, the concentrates joining the flotation concentrates in the filter plant, and the tailing going to waste. Further treatment of a middling product from these tables is contemplated and studies have been made along this line, but as yet nothing definite has been reached. The overflows from both the drag and the hydraulic classifiers are united and pumped back to the 16 re-treatment cells, the concentrate from which goes to the cleaners, and the tailing to waste. Throughout the mill adequate provision has been made for sampling.

Flow Sheet with the Inspiration Flotation Machines.—Except for a special arrangement in one unit, the only difference between this and the Callow flow sheet is in the flotation department.

From a metallurgical standpoint the difference here is not radical. In a few words, the essential difference can be stated as follows:

The Inspiration system makes use of but two machines per unit, a rougher and a cleaner placed end to end (Fig. 9). These are each designed to allow a free passage of the feed from the upper or feed end, to the tailing or discharge end, during which passage the feed is subjected to the frothing action of several compartments. Because of the large number of compartments (16 for the rougher), it has been found that a tailing can be made, the slime of which does not require re-treatment as is the case in the Callow system. The sand contained in the rougher tailing is subjected to table concentration. The tailing from the cleaners is pumped back to the roughers for a second treatment.

Several mechanical features have been introduced which are expected to keep the cost of operation at a minimum. The arrangement is simple,

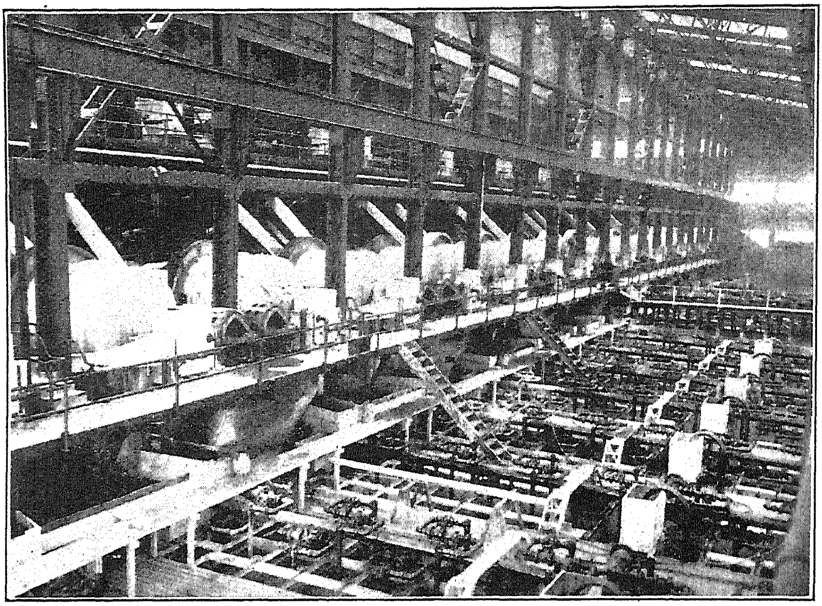

FIG. 8.—BALL MILLS AND FLOTATION MACHINES.

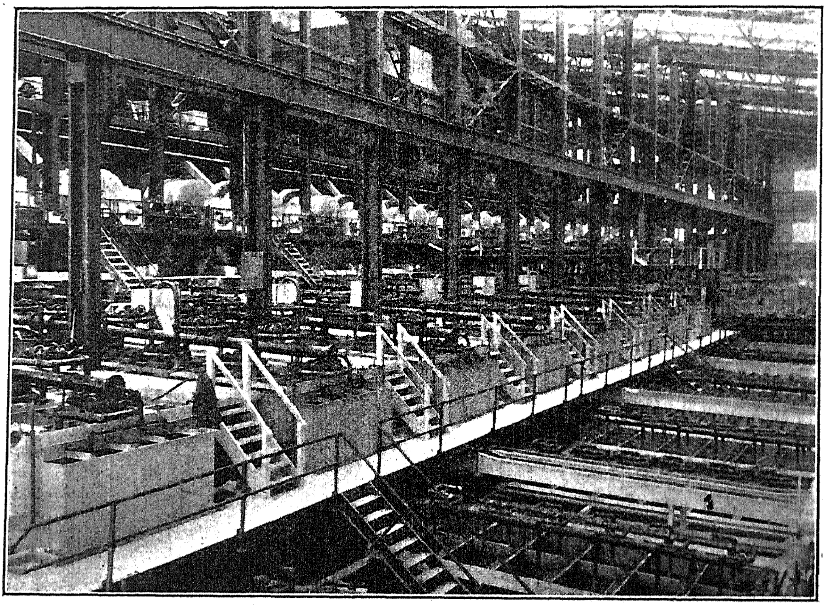

FIG. 9.—INSPIRATION FLOTATION MACHINES.

requires a minimum of floor space and is easily attended. Its large capacity is an important feature.

Flotation Air Supply.—The flotation department requires a large volume of low-pressure air. This is supplied by four single-stage centrifugal compressors occupying one end of the flotation floor. The rating of these compressors is as follows:

Capacity, 23,000 cu. ft. of inlet air per minute.

Discharge pressure, 5.75 lb. per square inch.

Revolutions per minute of impellers, 3,850.

Horsepower of driving motor, 720.

The step-up to the impellers is made through Alquist gears. One unit of the four is held in reserve.

Concentrates Filter Plant

Both the flotation and table concentrates receive the same treatment. From the concentrator they are either elevated or flow by gravity into the five 60-ft. and three 80-ft. Dorr tanks, located on either side of the elevator house. Here the moisture content is reduced to about 50 per cent. The thickened product is drawn off through the bottoms into launders and conveyed through tunnels to bucket elevators, which raise it to the top of the elevator house where it is distributed to six Oliver filters (Fig. 5).

The overflow from the Dorr tanks is practically clear and flows to the return-water sump. The filters are standard Oliver filters, the drums being 11 ft. 6 in. in diameter and having a length of 12 ft. After leaving the filters the concentrates have a moisture content of about 17 per cent. The arrangement is such that a single 20-in. conveyor belt traveling at 100 ft. per minute takes the concentrates away from the filters and delivers them to a railroad bin of the tank type. Here they are loaded into hopper-bottomed steel cars of 60 tons capacity. A 350-ft. train shed protects the loaded cars from heavy rains and winds.

Drag Classifiers

The drag classifier used on the flotation tailings for separating sand from slime is the result of experimental work in the test mill. It is worthy of a detailed description not alone on account of its ability as a classifier but also because of its large capacity. Except for the concentrates taken out by the flotation cells, each machine handles the entire tonnage of one unit, or 800 tons per day.

The classifier consists essentially of two 18-in. belts, running parallel to each other and so arranged that the return belts run entirely clear of the settling surface, the rough classification being carried on sufficiently

far below the surface to cause no disturbing currents. This is made possible by the use of four pulleys, one at the discharge end, two (one vertically above the other) at the feed end, and one at the center near the bottom, where the slope begins. The distance between head and tail pulleys is 39 ft. The direction of travel of the return belts is horizontal,

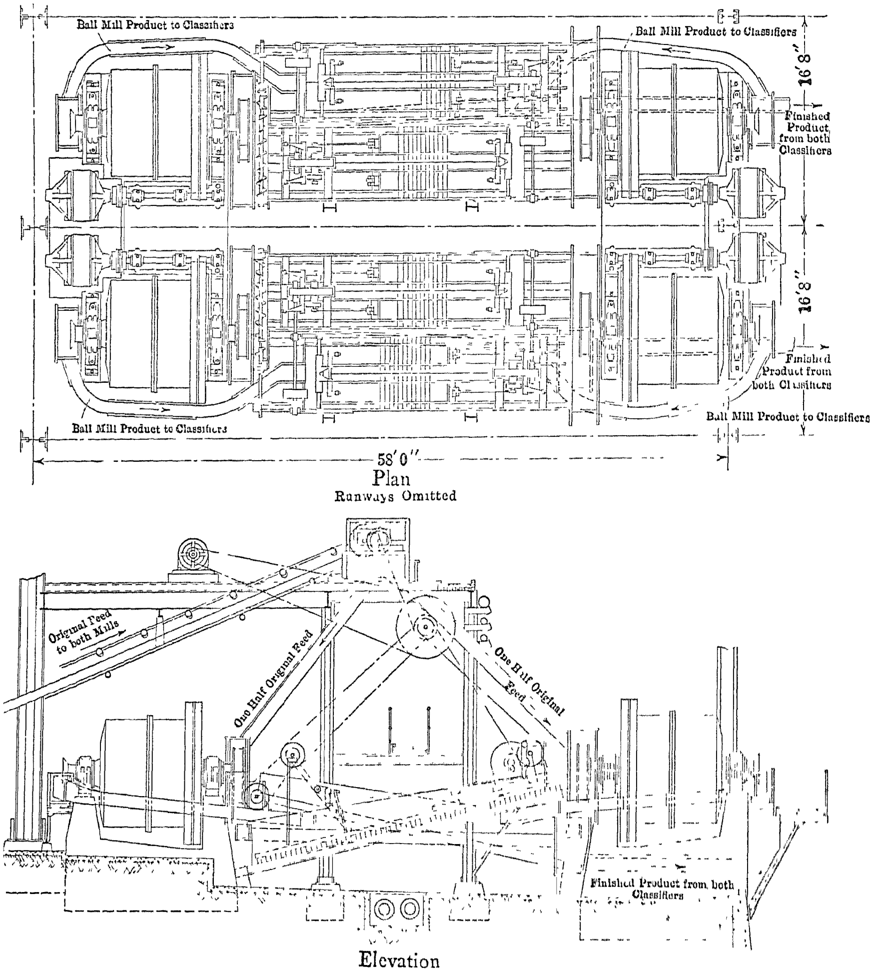

Fig. 10.—Ball Mill and Classifier Arrangement.

but on passing over the upper tail pulley it becomes vertical. The belt thus enters the pulp perpendicular to its surface. At the bottom of the tank its direction is again changed to the horizontal by the lower tail pulley. It travels thus to the center of the tank where a break is made; the belt with its load passes under another pulley and starts up a 20° slope toward the discharge end. All pulleys are 22 in. in di

Two adjustable overflow launders on either side of the tank having a combined lip length of 52 ft., carry off the slime. The effective settling area of the tank is 25 ft. by 4 ft., or 100 sq. ft. The overflow is 5 ft. above the bottom horizontal belt. The two sets of submerged pulleys are mounted on shafts running in water-tight bearings supported by the sides of the tank.

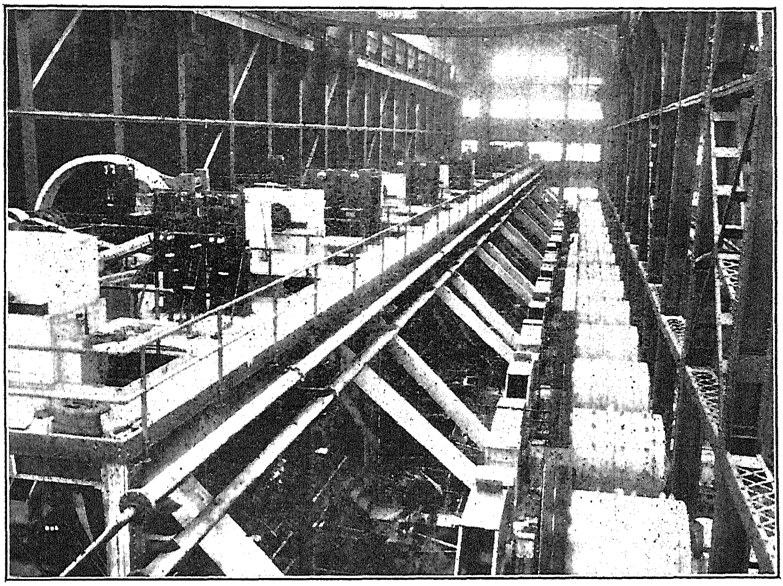

FIG. 11.—BALL MILLS AND MOTOR PLATFORMS.

Tailing Settling System

The conservation of the water supply is imperative, and it was for this reason that the extensive tailing-settling and return-water system was installed. The water is reclaimed at the concentrator and at the tailing dams. The equipment at the concentrator consists of three 100-ft. and one 200-ft. Dorr thickeners, and a large dewatering box 17 ft. by 109 ft. The overflow from the 80-ft. concentrate tanks is also turned into the return system. The dewatering box handles only the table tailing, the overflow from it being further settled in three of the 60-ft. Dorr thickeners. The overflow from all tanks goes to a 210,000-gal. concrete sump from which it is pumped to a 60-ft. diameter steel tank of like capacity on the shop level above the concentrator.

The return-water pump equipment consists of four vertical triplex pumps direct-driven through gearing by 100-hp. synchronous motors. These pumps each have a capacity of 2,000 gal. per minute and work against a head of 113 ft.

The water reclaimed from behind the dam (Fig. 12) is at present being handled by a 3,000-gal. two-stage centrifugal pump, so arranged that it can be moved up a skidway as the water level of the pond behind the dam rises.

200-Ft. Dorr Thickener.—The 200-ft. Dorr thickener is larger than any single unit ever installed and is, therefore, interesting from a mechanical standpoint. The thickener has not yet been erected but at this time the designs are complete. It consists essentially of a reinforced-concrete tank 200 ft. in diameter with the bottom sloping toward the center and having a depth at the outside of 7 ft. 3 in. The feed is delivered at the center of the tank through a launder supported by a 107-ft. steel truss. At the center of the tank is a steel plate pivot, 18 ft. high, upon which is

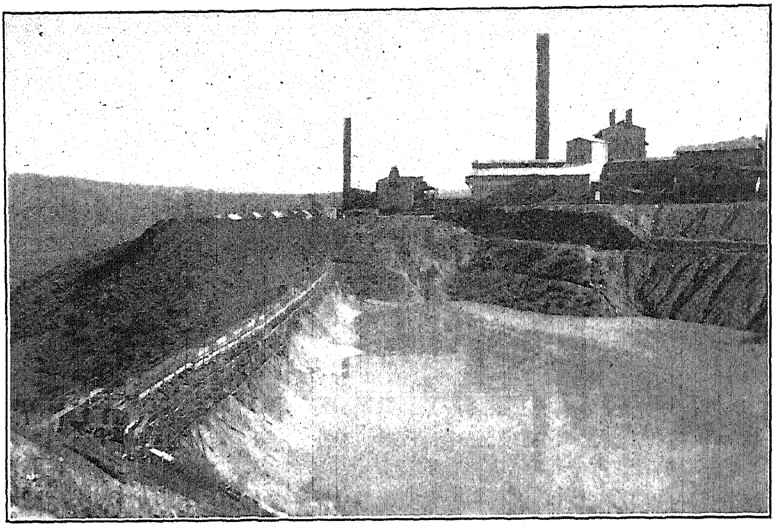

FIG. 12.—TAILINGS DAM.

mounted the drive drum. To this drum are fastened four short rakes (two 24 ft., and two 35 ft.) and the driving truss, which in turn carries two long rakes, 10 ft. apart and 100 ft. in length. The slope of the bottom of the tank varies, it being 2$\frac{3}{16}$ in. per foot near the center, 1$\frac{3}{4}$ in. per foot for the short rakes and the inner 16 ft. of the long rakes, and the remainder 1 in. per foot. The driving truss is completely submerged except for the outer end, which rests on a circular track on the tank wall. The driving mechanism is placed at the outer end and consists of a 5-hp. motor which is connected through gearing to a single 24-in. plain-tread driver. Two similar wheels are placed 10 ft. on either side of the driver to distribute the weight. The weight of the driving truss provides the necessary traction. The thickened product is drawn off through an

annular opening at the center, and the overflow at two diametrically opposite points on the periphery of the tank.

GENERAL EQUIPMENT AND SUPPLIES

Water Supply

Various possibilities were presented for supplying water to this plant. To insure a supply, sufficient ground and water rights were early purchased at Wheatfields, about 12 miles from the mill site and 1,000 ft. lower in elevation. Before developing this supply, however, it was decided to prospect on the flat below the tailing storage site. Wells were sunk at various points until the best location had been determined, which is 2½ miles from the mill, 430 ft. lower and at the junction of two fair-sized drainage channels, receiving their supply from the Pinal Mountains about 10 miles away and 4,000 ft. higher. Here six wells have been sunk and 24-in. multi-stage turbine well pumps installed. These deliver through wooden pipe lines to a common steel sump tank having a capacity of 235,000 gal. Each well pump is belt-driven by a 150-hp. vertical motor and delivers a maximum of 1,200 gal. per minute. Near the sump tank is located the pumping station which contains six 1,200-gal. pumps delivering into a common 20-in. pipe line. These pumps are horizontal, duplex, double-acting and are direct-driven through herringbone gears by 300-hp. synchronous motors taking current at 6,600 volts. Power for the pumping station and wells is supplied by the Inspiration-International power house. A 10-ton crane serves all parts of the building.

The 20-in. pipe line is 14,600 ft. long with a rise of 520 ft., and delivers water to the storage reservoir near the concentrator. From this reservoir water is delivered to all parts of the property. It is located about 80 ft. above the concentrator, 1,200 ft. away, and has a capacity of 3,000,000 gal. An oval excavation about 20 ft. deep was made in the top of a hill and the sloping sides and bottom lined with concrete. Two 14-in. pipe lines deliver the water to the concentrator.

Power Supply

Electric power is obtained from two sources. The Reclamation Service of the United States Government, from its hydro-electric plant at the Roosevelt Dam, 40 miles away, furnishes energy at 40,000 volts, 25 cycles, three-phase.

In conjunction with the International Smelting Co., the Inspiration company built a power house that utilizes the waste heat from the reverberatories under one set of boilers. Another set of boilers, oil-fired, is also used when the power demand is greater than can be supplied by the waste-heat boilers alone. This power house contains three 7,500-kva.

steam-turbine-driven generators, delivering energy at 6,600 volts, 25 cycles, three-phase; also three 15-lb. reciprocating blowing engines, each having a capacity of 15,000 cu. ft. of free air per minute. The air is for the smelter use exclusively, while the electric energy is for the mining, smelting and ore-dressing plants.

The distributing system is so laid out that any division of the plant can take power from either source. When available, the Reclamation power is used for the greater part of the Inspiration load. Each unit of the plant has its transformer station for stepping down the transmission line voltage.

Motors of 50 hp. or greater are run on 2,200 volts, smaller motors on 440 volts, and lighting circuits on 110 volts. Synchronous motors are used as required to maintain a satisfactory power factor.

Transformer Stations

There are two separate outdoor transformer stations, one at the concentrator and one at the mine plant. These stations are of rather unusual design, being skeleton-steel structures. The supports are of pipe poles and the trusses between which the busses are stretched are angle lattice construction.

As the power from the two sources is received at different voltages each station requires two sets of transformers. The Reclamation energy is delivered at 40,000 volts and the power house at 6,600 volts, both being stepped down to 2,200 volts for distribution about the plant.

Both high-tension transmission lines are in duplicate circuits. The incoming lines pass over electrolytic lightning arresters to electrically operated remote-controlled circuit-breakers, then through electrically operated remote-controlled automatic circuit-breakers to the transformers. Any one transformer may be cut in or out as desired, or all at one station can work in parallel. At the concentrator there are eight outdoor-type, 2,000-kva., oil-insulated, water-cooled, three-phase transformers, four on the 40,000-volt line and four on the 6,600-volt line. At the mine there are four similar transformers, two for each incoming line.

The 2,200-volt circuits are led from the transformers to the distributing stations, the one at the mine being located in the compressor and hoist house and the one at the concentrator in a separate building. Each distributing station consists of a concrete cell structure, carrying two sets of 2,200-volt busbars, one for each power line. This cell structure also carries the disconnecting switches, meter transformers and oil switches as required by the remote-control feature of the switchboard. In front of the cells is the switchboard with separate panels for the control of each incoming high-tension line, each set of transformers and each outgoing 2,200-volt circuit. Wattmeters, frequency meters, and power-

factor meters, all graphic, and integrating watt-hour meters are installed for each power supply, and ammeters, watt-hour meters and graphic wattmeters for each outgoing circuit. Two-throw oil switches, controlled from the switchboard, make it possible to run any circuit on either power, or to disconnect it from both.

The transformer and distributing stations, besides presenting an attractive appearance, are unusually complete, convenient for the attendant and provide the greatest possible safety.

Shops and Railroad Facilities

All shop work, except that for the mine, is done at the concentrator shops where a complete equipment is installed. The equipment comprises about 30 different machine tools, all with individual motor drives, a 40-ton electric crane, traveling the entire length of the building, the necessary facilities for locomotive repairs, several welding outfits for various classes of work and a forge shop equipped with both oil and coal forges. The warehouse and electric shop and the locomotive roundhouse occupy opposite ends of the shop building. A standard-gage track through the center of the building permits cars to be run either into the shop or through the shops into the warehouse.

The shop level is 22 ft. above the upper floor of the concentrator. Communication between the two buildings is provided by the inclined skipway which occupies one end of the concentrator. This connects with the shops by a well through which material can be transferred either into the shops from the mill for repairs, or into the mill from railroad cars.

The railroad facilities comprise the main line and several spurs, the total length being about 10 miles. All equipment is standard gage. The lines connect the several units of the surface plant, except the pumping plant and the Live Oak Division. These are reached via the Arizona Eastern. On account of the topographical conditions, it was necessary to use switchbacks to make the grade between the Arizona Eastern on the flat and the concentrator, about 400 ft. higher. The switchbacks provide for handling six cars. The extremes of grade and curvature on this line are 4 per cent. and 12° respectively. The line which connects the mine and the mill has a grade of 0.3 per cent. in favor of the load.

The six-wheel, side-tank type of locomotive is used. Two are used on the mine line. These have a weight on the drivers when loaded of 112,000 lb. A third locomotive is used on the low line, which has a corresponding weight of 200,000 lb.; the latter being equipped with a superheater and Walschaerts' valve gears, oil burners being used on all. The ore cars are of the Ingoldsby patent-dump type, have a capacity of 60 tons and a train length of 33 ft. They are of all-steel construction and weigh about 22 tons each.

Electric haulage on the mine division was at one time considered, but for sake of uniformity on the several lines the idea was abandoned.

CONCLUSION

The salient features of the mine plant can be summarized as follows: The dual arrangement of the entire plant; the automatic loading and hoisting of ore; the application of the elevator principle for handling men; and the reduction of the crude ore for mill treatment in two simple stages.

The final product of the plant is ore in storage ready for mill treatment. It carries roughly $1\frac{3}{4}$ per cent. copper, mainly sulphides, and has been crushed to pass a 2-in. opening of the disk crushers. The average moisture content is $2\frac{1}{2}$ per cent.

To the mine summary can be added the following covering the remainder of the plant: A present mill recovery on sulphides of 90 per cent., or 85 per cent. on the total content; finished grinding in one machine; and flotation followed by table concentration.

Although the Inspiration plant is not quite the largest, it does embody what is today the latest practice in the treatment of low-grade chalcocite ores. In certain departments considerable pioneering has been done, which has resulted in marked economies, advantage of which will no doubt be taken where similar conditions make them applicable.

The Decomposition and Reduction of Lead Sulphate at Elevated Temperatures

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Edited by H. O. Hofman,† Boston, Mass.

(Arizona Meeting, September, 1916)

I. *Introductory*

LEAD sulphate occurs as anglesite, and is formed in every roasting of lead sulphides or sulpho-salts containing lead. In smelting in the blast furnace an ore containing natural or artificial lead sulphate, a large part of the sulphur present goes to the formation of the intermediary product matte which requires further treatment for the recovery of the values it contains. It is therefore of importance to study the behavior of lead sulphate at elevated temperatures, both without and with reducing agents, especially as the information available gives data which are more or less conflicting.

II. *Decomposition of $PbSO_4$ by Heat*

The dissociation of $PbSO_4$ into PbO and SO_3 or SO_2 and O has been discussed by many metallurgists. The following data are considered to be representative. Doeltz and Graumann[1] found that $PbSO_4$ was not affected by heat at 800° C.; that dissociation began at 900° C., and was rapid at 1,000° when the salt lost in 1½ hr. 14.3 per cent. of its weight, the dry salt containing 26.43 per cent. SO_3. Hofman and Wanjukow[2] summarize their work upon the dissociation of metallic sulphates with the statement that the decomposition of $PbSO_4$ begins at 195° C., but progresses very slowly, with the formation of the basic salt $6PbO.5SO_3$ at 705°; that this salt undergoes a transformation at 847°, begins to be decomposed at 888°, to sinter at 896°, and to fuse at 910°; rapid dissociation begins at 952°, and is accompanied by volatilization of PbO; retardations at 958° and 962° indicate an acceleration of decomposition which, however, is not complete. Proske[3] found that $PbSO_4$ gave off at 900° C. in 30 min. only 0.63 per cent. of its SO_3, and that the loss increased with

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rise of temperature exceeding 100 per cent. at 1,100°, a figure which showed that PbO was volatilized at this temperature in a current of air. Schenck and Rassbach[4] observe that $PbSO_4$ melts above 1,100° C.; Calcagni and Marotta[5] give the melting point as lying between 1,000° and 1,010° C.

The lead used for the preparation of $PbSO_4$ was the chemically pure metal of Kahlbaum. It was dissolved in HNO_3, brought to crystallization, the crystals of $Pb(NO_3)_2$ were recrystallized, dissolved and treated with $(NH_4)_2SO_4$; the $PbSO_4$ formed was filtered, washed, dried, and heated to 750° to 780° C., preliminary tests having shown that no dissociation took place below 800° C. Wet assays by the Alexander molybdate method[6] gave the following results:

$PbSO_4$, mg		Pb, mg	Theoretical Content Pb, mg
250	gave	174 7	174 2
253	gave	173 5	172 9
282	gave	192 8	192 9

The $PbSO_4$ was therefore pure, the difference between the actual and theoretical contents of lead lying within the limit of error of the assay method.

In the heating experiments, about 1 g. of $PbSO_4$ was used. The substance was placed in the platinum boat, and this in a boat of unglazed porcelain. For temperatures up to 550° C. a glass tube wound with nickel wire 0.5 mm. in diameter formed the heating apparatus, a current of from 4 to 5 amp. giving a temperature of 600° C. The glass tube was placed in the clay cylinder, and the latter on supports in a muffle. Readings were taken with a mercury thermometer. For temperatures above 550° C. a horizontal Heraeus platinum-wound furnace was employed; the boats with the substance were placed in the silica tube, 65 cm. long and 20 mm. in diameter, which was surrounded by the clay tube. The zone of constant temperature was from 5 to 6 cm. long. The temperatures were measured with a thermocouple and the readings taken with a Siemens-Halske millivoltmeter. A solution of $BaCl_2$ acidified with HCl, to which had been added some Br, formed the reagent for the detection of escaping SO_2 and SO_3; it was sensitive to 0.2 mg. SO_3. In all cases a current of air, purified and dried with $KMnO_4$, KOH, concentrated H_2SO_4, and a $CaCl_2$ tower, was passed at a uniform speed through the heating furnace.

In both series of experiments losses in weight formed the criteria of occurring changes. In Table I are recorded three series of experiments covering a range of temperature from 200° to 800° C., inclusive. The greatest loss in weight including 775° C. was 0.68 per cent. As there was no indication whatever of escaping SO_3 in the $BaCl_2$ solution, the loss must be attributed to the release of moisture contained in the sample.

In the subsequent tests the sample was always heated to 750° C. before a weight was taken in order to be sure that all inclosed water had been driven off.

The dissociation of $PbSO_4$ which, as shown by Table I, begins at 800° C., was carried further to 1,000° C. The results of two series of tests are given in Table II. This shows (1) that the partial dissociation begins at 800° C.; (2) that the speed is slow up to 950°, and quick when the temperature is raised above 950°; (3) that fusion, which is seen between 950° and 1,000°, takes place only after a partial decomposition of the normal $PbSO_4$.

TABLE I.—*Heating of $PbSO_4$ from 200° to 800° C.*

Test No.	$PbSO_4$ before Heating, Milligrams	Temperature, Degrees C.	Time of Heating, Hours	$PbSO_4$ after Heating, Milligrams	Loss in Weight		Remarks
					Milligrams	Per Cent	
				Series I			
1	1,202.0	200	1.0	1,198.0	4.0	0 33	
1*a*	1,198.0	200	4 0	1,198.0			
2	1,198 0	300	0 5	1,197 2	0 8	0.07	
2*a*	1,197 2	300	1.0	1,197.2			
3	1,197.2	400	0 5	1,196.2	1.0	0 08	
3*a*	1,196.2	400	2 5	1,196 2			
4	1,196 2	500	0.5	1,196.2			
4*a*	1,196.2	500	0.5	1,196 2			
5	1,196.2	600	0.5	1,196.2			
5*a*	1,196 2	600	0.5	1,196 2			
6	1,196.2	700	0.5	1,196.2			
6*a*	1,196.2	700	0.5	1,196.2			
7	1,196.2	800	1.0	1,195.6	0 6	0.05	Evolution of SO_3 begins.
				Series II			
8	1,190.0	750	0.5	1,184.6	5.2	0.44	
8*a*	1,184.6	750	0 5	1,184.6			
9	1,184.6	780	0.5	1,184.6			
10	1,184.6	800	0.5	1,184.4	0.2	0.01	Evolution of SO_3 begins.
10*a*	1,184.4	800	0.5	1,183.9	0.5	0.04	Evolution of SO_3 continues.
				Series III			
11	500.0	775	1.5	496.7	3.3	0 66	No evolution of SO_3.
11*a*	496.7	775	4.5	496.7	...	....	
11*b*	496.7	775	7.0	496.7	...	....	
12	500.0	775	1 5	496.6	3.4	0.68	
12*a*	496.6	775	4.5	496.6	...	....	
12*b*	496.6	775	7 0	496.6	...	....	

TABLE II.—*Heatings of $PbSO_4$ from 800° to 1,000° C.*

Test No	$PbSO_4$, Dried at 750° C before Heating, Milligrams	SO_3, Milligrams	Temperature, Degrees C.	Time of Heating, Hours	$PbSO_4$ after Heating, Milligrams	Loss in SO_3		Remarks
						Milligrams	Per Cent	
				Series I				
1	1,152.0	304 3	800	1.0	1,151 0	1.0	0.33	
1a	1,151.0	304.3	850	1 0	1,147 3	3 2	1 05	Evolution of SO_3.
2	1,024 6	270.7	900	1 0	1,011.5	13 1	4 48	
3	1,192.2	314.8	950	1 0	1,149 8	42 4	13 46	Charge sinters.
4	1,129 5	298.1	1,000	1 0	978 4	151 1	50.30	Charge fuses.
4a	Heating continued	298.1	1,000	0 5	......	24 6	.....	SO_3 is still evolved.
				Series II				
5	1,000 4	264 0	800	1 0	999 8	0.6	0.23	
6	1,000 0	264 0	850	1.0	997 8	2 2	0 83	Evolution of SO_3.
7	1,000 0	264 0	900	1.0	989 6	10.4	4 00	
8	1,000.4	264 0	950	1 0	965 8	34.6	13 10	
9	1,000 2	264 0	975	1 0	936.0	65.2	24 70	Charge partly fused.
10	1,000 2	264 0	1,000	1.0	878 5	121 7	46 00	Charge fused.
10a	1,000 4	264 0	1,000	1.0	879 4	121.0	46 00	Charge fused.

Up to 1,000° C. no volatilization of PbO was observed; above 1,000° it was very noticeable, forming a sublimate on the boat. Pure PbO, it will be remembered, begins to be volatilized at 700° C., and is decidedly volatile at 800°.[7]

III. *Decomposition of $PbSO_4$ by Heat in the Presence of SiO_2*

The process of slag-roasting galena ore is based upon the decomposition of $PbSO_4$ by SiO_2 at an elevated temperature with the formation of lead silicate. The sulphur content may be reduced thereby to about 1 per cent. In blast roasting, SiO_2 acts in a similar manner in the decomposition of $PbSO_4$; in addition, it acts at a low temperature as an inert substance holding apart the particles of galena that they may roast more readily, and at a high temperature it slags the PbO formed in the roast, which, in its turn, scorifies the other components of the charge.

In a study of the decomposition of $PbSO_4$ by SiO_2, it is first necessary to consider the union of PbO and SiO_2. Mostowitsch[8] has shown that mixtures of PbO and SiO_2 ranging from $6PbO.SiO_2$ to $PbO.SiO_2$ begin to soften at temperatures lying between 700° and 750° C., *i.e.*, considerably below the melting point of PbO which lies at 883° C.; also that the melting points of lead-silica glasses are lowered as the SiO_2 contents are raised.

Thus the subsilicate, $4PbO.SiO_2$, is completely liquified at 726°; the singulosilicate, $2PbO.SiO_2$, forms a viscous liquid at 724° and requires 940° C. to flow readily. The low temperature range of 700° to 730° C. in which PbO and SiO_2 begin to combine makes its probable that SiO_2 will greatly aid the decomposition of $PbSO_4$ and cause it to take place at a low temperature.

The only research dealing with the action of SiO_2 upon $PbSO_4$, which gives numerical data, is that by Hilpert.[9] He found that the decomposition of $PbSO_4$ began at 720° C., that the $PbSO_4$ in the bisilicate mixture —$PbSO_4 : SiO_2$—held for half an hour at 810° C., was decomposed to the extent of 10 per cent.; at 850° to 20 per cent.; and at 900° to 100 per cent. The data obtained in the present investigation differ to a considerable degree from those of Hilpert.

In the experiments to be described, silica containing 99.9 per cent. SiO_2 served as raw material; the triturated mixtures tested covered a range extending from the subsilicate to the trisilicate; the amount of $PbSO_4$, dried at 750° C., was kept constant. In the preliminary work it was noticed that variations in the rate of flow of the purified and dried air through the tube furnace caused irregularities in the results. The volume of air and the pressure of the current were therefore kept constant by using a gasometer of 1,000 c.c. capacity and a regulator. Each heating lasted 30 min.

The results are assembled in Table III. They show (1) that the dissociation temperature of $PbSO_4$ is not lowered by the presence of SiO_2; (2) that decomposition of $PbSO_4$ by SiO_2 becomes marked at 950°, and increases as the temperature is raised. The percentage losses in excess of 100 per cent. at 1,100° show that PbO has been volatilized; (3) that the decomposing effect of SiO_2 is not proportional with the SiO_2 content of the mixture, and that rather the reverse holds true. Decomposition appears to be governed by the viscosity of the lead silicate which is formed; in the bi- and trisilicate mixtures viscous slag envelopes $PbSO_4$ and retards action. The most rapid decomposition accompanied by the lowest loss of lead by volatilization is found with a silication lying between the singulo- and the bisilicate and containing from 10 to 15 per cent. SiO_2.

In applying the foregoing results to blast roasting, the Savelsberg process is chosen as a type, as it treats charges made up of raw materials and permits little latitude in varying the SiO_2 content. In preparing a mixture for blast roasting, the sizes of particles and the chemical analyses of the constituents have to be taken into consideration. Leaving aside the grain size, the variations permissible in the percentages of Pb, Fe, S, and CaO are large in comparison with those of SiO_2, as the SiO_2 content is confined within narrow limits. Practical experience with the Savelsberg process has shown that with a very basic charge, the pots run hot, the sulphur content is easily reduced to 1.5 per cent., and the amount

TABLE III.—*Heatings of $PbSO_4$ with Varying Amounts of SiO_2*

Test No.	$PbSO_4+SiO_2$, Dried at 750°C. before Heating, Milligrams	$PbSO_4$, Milligrams	SO_3, Milligrams	Temperature, Degrees C	Time of Heating Hours	$PbSO_4+SiO_2$ after Heating, Milligrams	Loss in SO_3 Milligrams	Loss in SO_3 Per Cent.	Remarks
	Series I: Subsilicate, 4 $PbSO_4$:SiO_2								
1	520.9	496.6	131.2	760	2.0	520.9			
2	520.9	496.6	131.2	800	0.5	520.9			
3	520.9	496.6	131.2	850	0.5	520.9	0.7	0.53	Sintering starts
4	521.2	496.6	131.2	900	0.5	515.8	5.4	4.11	
5	519.8	496.6	131.2	950	0.5	496.0	23.8	18.14	
6	520.4	496.6	131.2	975	0.5	483.7	36.7	28.00	Fusion starts
7	520.5	496.6	131.2	1,000	0.5	464.5	56.0	42.68	Fusion complete.
8	521.5	496.6	131.2	1,050	0.5	415.2	106.3	81.02	
9	522.8	498.0	131.5	1,100	0.5	349.8	173.0	131.05	
	Series II: Singulosilicate, 2$PbSO_4$:SiO_2								
10	544.0	496.6	131.2	800	0.5	543.8	0.2	0.15	
11	543.8	496.6	131.2	850	0.5	543.0	0.8	0.61	Sintering starts
12	544.2	496.6	131.2	900	0.5	538.8	5.4	4.11	
13	545.0	496.6	131.2	950	0.5	516.6	28.4	21.64	
14	544.6	496.6	131.2	975	0.5	503.8	40.8	31.10	Fusion starts
15	544.6	496.6	131.2	1,000	0.5	487.6	57.0	43.50	Fusion partial
16	549.0	500.0	132.0	1,050	0.5	438.6	110.4	83.63	
17	547.6	498.0	131.5	1,100	0.5	390.8	156.8	118.80	
	Series III: Bisilicate, PbO:SiO_2								
18	593.5	496.6	131.2	800	0.5	593.3	0.2	0.15	
19	593.6	496.6	131.2	850	0.5	592.6	1.0	0.75	
20	594.0	496.6	131.2	900	0.5	589.0	5.0	3.75	Sintering starts
21	594.8	496.6	131.2	950	0.5	564.5	29.4	22.44	
22	594.0	496.6	131.2	975	0.5	546.7	47.3	36.10	
23	594.5	496.6	131.2	1,000	0.5	527.8	66.7	50.84	Fusion starts
24	598.0	499.2	131.7	1,050	0.5	488.0	110.0	83.52	
25	596.8	498.0	131.5	1,100	0.5	459.4	137.4	104.48	
	Series IV: Trisilicate, 2$PbSO_4$:3SiO_2								
26	643.0	496.6	131.2	800	0.5	642.6	0.4	0.30	
27	642.6	496.6	131.2	850	0.5	641.6	1.0	0.75	Sintering starts.
28	643.2	496.6	131.2	900	0.5	637.4	5.8	4.35	
29	643.2	496.6	131.2	950	0.5	617.8	25.4	19.36	
30	645.6	496.6	131.2	975	0.5	592.8	52.8	40.24	
31	643.4	496.6	131.2	1,000	0.5	565.7	77.7	59.30	Fusion starts.
32	649.2	500.0	132.0	1,050	0.5	528.8	120.4	91.21	
33	649.2	500.0	132.0	1,100	0.5	521.2	128.0	97.00	

of unagglomerated fines is large. These cause a large loss of lead by volatilization owing to the metallic lead set free by the actions of $PbSO_4$ and PbO upon PbS. The blast-roasted cake is permeated by pellets and veinlets of metallic lead which make it tough and difficult to break;

it also contains considerable amounts of uncombined PbO which makes dust, causes loss, and injures the men. With a very acid charge, on the other hand, there occurs premature sintering or fusing which prevents the blast from penetrating satisfactorily and gives a product running high in sulphur, 5 per cent. and over, mainly, as sulphide; there is formed only a small amount of fines, the cake is readily broken, and makes little dust. The slagged material is difficult of reduction in the blast furnace and is likely to cause slags to run high in lead.

IV. *Reduction of $PbSO_4$ by CO*

In the tests to be described, the CO gas was generated by the action of concentrated H_2SO_4 upon C H_2O_2; it was collected over water in a holder and cleaned by passing through KOH, H_2SO_4, and a $CaCl_2$-tower. The $PbSO_4$, dried in an air current, as in the preceding tests, was held at different temperatures for 1 hr. in a current of CO after the air in the tube furnace had been expelled by the gas. During this interval of time from 1.5 to 1.6 liters of CO passed over the salt. The escaping gas was passed through a wash-bottle containing $BaCl_2$, HCl, and Br. The results are given in Table IV.

TABLE IV.—*Reduction of $PbSO_4$ by CO*

Test No	$PbSO_4$, Dried at 750° C before Heating, Milligrams	Oxygen Content		Temperature, Degrees C.	$PbSO_4$+PbS +Pb, Heating, Milligrams	Loss in weight		$PbSO_4$ Unchanged after Heating, Milligrams	$PbSO_4$ Reduced after Heating	
		Milligrams	Per Cent.			Milligrams	Per Cent		Milligrams	Per Cent
1	382 7	80 79	21 11	525	382.7					
2	553 6	116 87	21 11	525	553.4	0 2				
3	753 8	159 13	21 11	550	753 8					
4	756 2	159.64	21.11	600	755 1	1 1	0.15	749 0	7.6	0.8
5	753 8	159 13	21 11	600	753 4	0 4	0 05	749.3	4.5	
6	759.0	160 23	21 11	630	595 2	163.8	21 58	194.4	564 6	74.4
7	283 6	59 87	21.11	650	213 1	70 5	24 86			
8	758 2	160 06	21.11	650	567 5	190.7	25.15	16.44	741.8	97.84
9	753 6	159 10	21.11	700	558 6	195 0	25 87	28 47	725.1	96 22
10	439 5	92 78	21 11	770	320.5	119.0	27 07			

They show that the reduction of $PbSO_4$ by CO begins at 600° C.; the salt loses in weight and becomes dark; the $BaCl_2$-solution becomes clouded, proving that SO_2 is set free. At 630° C. a heavy precipitate forms in the $BaCl_2$ solution, the salt becomes dark, there are visible pellets of lead, and the loss in weight of 21.58 per cent. exceeds the oxygen content, which is 21.11 per cent. All the phenomena prove that a reaction between undecomposed $PbSO_4$ and reduced PbS is taking place according to the two equations: $PbSO_4 + PbS = 2Pb + 2SO_2$ and $3PbSO_4 + PbS = 4PbO + 4SO_2$;[10] the PbO formed in the second reaction is reduced to Pb by CO. This reduction takes place according to Brislee[11] at 300° C. with the forma-

tion of Pb_2O, while Schlagdenhauffen and Pagel[12] give 430° C. as the temperature at which Pb is formed. The experiments show that $PbSO_4$ can not be reduced completely to PbS by CO because the reduction is accompanied by a reaction between $PbSO_4$ and PbS.

In order to test the figures of Brislee and of Schlagdenhauffen and Pagel the experiments shown in the accompanying table were carried out. Litharge was heated in a current of CO gas for periods of 40 min. to temperatures ranging between 400° and 500° C. The degree of reduction was ascertained by the loss in weight. The table shows that at 400° C. only Pb_2O is formed; the product has a black color, and a satin luster; it is decomposed by HNO_3 and $C_2H_4O_2$ into lead acetate and spongy lead. Between 450° and 500° the product consisted of Pb with some Pb_2O; at 500° there was still present a small amount of Pb_2O. The reduction of PbO by CO may be expressed by: PbO→300° C.→Pb_2O→> 430° C.→Pb.

Reduction of PbO by CO

Test No.	PbO before Heating, Milligrams	Oxygen Content, Milligrams	Temperature, Degrees C.	PbO after Heating, Milligrams	Loss in Weight, Oxygen Expelled, Milligrams	Loss in Oxygen by Reduction, Per Cent.	Remarks
1	300.6	10 78*	400	292 2	8 4	77 93	No metallic lead is visible.
2	245.0	8.78*	400	238 0	7 0	79 68	
3	281.0	20.15	450	264 2	16 8	83 37	Metallic lead and Pb_2O.
4	289.2	20 74	500	270 4	18 8	90 65	
5	291 1	20 81	500	271 1	20 0	95 80	Metallic lead with little Pb_2O.

* These figures give in milligrams the oxygen of Pb_2O calculated from milligrams of PbO used.

V. *Reduction of $PbSO_4$ by Means of C*

In these tests $PbSO_4$ was intimately mixed with sugar carbon, previously heated in a closed crucible, in the proportion of $1PbSO_4$ to 4C, and then exposed to different temperatures in a current of pure nitrogen for periods of 1 hr. The nitrogen, taken from a holder, was purified, tests with sugar carbon showing that it had been completely freed from oxygen. The results obtained are given in Table V. They show that the reduction, which begins at 550° C., is accompanied by the liberation of SO_2. At 630° the reduction proceeds more quickly than at 550°, the loss in weight and the amount of barium precipitate are larger, and pellets of metallic lead are seen; the $PbSO_4$ has been largely changed to PbS. At 700° the reduction to PbS and Pb is completed.

A comparison of the effects of the two reducing agents shows that CO begins to act at 600° C., and carbon at 550°; that the reducing power is increased with the temperature, when CO acts more powerfully within

the given limits than carbon, thus, for example, at 630° the CO reduced 74.4 per cent. and C only 36.98 per cent. of the $PbSO_4$.

TABLE V.—*Reduction of $PbSO_4$ by Means of Carbon*

Test No.	$PbSO_4$, Milligrams	Carbon, Milligrams	Mixture before Heating, Milligrams	Temperature, Degrees C.	Mixture after Heating, Milligrams	Loss in Weight, Milligrams	$PbSO_4$ Unchanged after Heating, Milligrams	Reduction of $PbSO_4$, Per Cent.		Remarks
1	604.3	95.7	700.0	550	698.0	2.0	602.9	0.25	0.25	Evolution of SO_2.
2	604.6	95.8	700.4	550	697.4	3.0	602.9	0.25		
3	604.7	95.9	700.6	600	681.9	18.7	578.8	4.33	3.98	Evolution of SO_2.
4	605.2	95.8	701.0	600	683.7	17.3	583.8	3.62		
5	605.2	95.8	701.0	630	621.0	80.0	381.4		36.98	Metallic lead is present in the reduced mixture
6	604.3	95.7	700.0	650	542.2	157.8	140.2	76.80	77.42	
7	604.3	95.7	700.0	650	537.6	162.4	132.7	78.04		
8	604.0	96.0	700.0	700	518.8	181.2	55.77	90.76	91.02	
9	605.0	95.8	700.8	700	519.6	181.6	52.76	91.28		

In applying the foregoing results to reduction in the blast furnace, we have to consider the reducing agents and the temperatures. In the blast furnace the reducing agents are the solid carbon of the coke and the gaseous CO of the ascending gas current which travels at a rate of about 22 ft. per second.[13] The furnace charge receives from 12 to 15 per cent. coke. The gases passing from the throat of the furnace at a temperature of from 100° to 150° C. contain CO_2 15 to 21, CO 5.4 to 11.0, and oxygen 0.4 to 1.0 per cent. by volume; usually the ratio of $CO_2 : CO$ is as 3 : 1. The reduction of $PbSO_4$ by carbon and CO is energetic at 600° C., and with it takes place the action of $PbSO_4$ upon PbS; both reactions occur at a temperature below that of decomposition of $PbSO_4$ by heat alone. The elimination of sulphur, which may amount to from 25 to 40 per cent.,[14] is therefore largely due to the action of $PbSO_4$ upon PbS in the upper part of the furnace. An advantage in using blast-roasted lead ore as regards elimination of sulphur lies in the fact that the sulphur is present mainly as sulphate which is readily eliminated as SO_2 and does not increase the matte-fall. The CaO of the limestone added to the blast-roasting charge ($CaCO_3$ = per cent. S + 2 per cent.) is converted into $CaSO_4$, and this is either reduced at 800° C[15] to CaS and enters the slag as sulphide, or it is dissociated at 1,000° C. by SiO_2[16] and enters the slag as silicate. The SO_3 of $CaSO_4$ in blast-roasted ore has therefore as little influence upon the matte-fall as has the SO_3 of $PbSO_4$. Slags running high in FeO have been found to hold in solution as much as 7 per cent. S; thus two slags made at Stolberg, Rhenish Prussia, with 55 and 38 per cent. FeO contained 7.4 and 3.7 per cent. S.

[1] Zur Bildung von Flugstaub und Ofenbruch im Bleihüttenbetriebe, *Metallurgie*, vol. 3, No. 13, p. 441 (July 8, 1906).

[2] The Decomposition of Metallic Sulphates at Elevated Temperatures in a Current of Dry Air, *Trans.*, vol. 43, 546 (1912).

[3] The Decomposition of Lead Sulphate by Ferric Oxide, *Metall und Erz*, vol. 1, No. 14, pp. 415 to 419 (Apr. 22, 1913).

[4] R. Schenck: *Physikalische Chemie der Metalle*, Knapp, Halle, 1909, p 178.

[5] *Gazzetta Chimica Italiana*, vol. 42, part 2, p. 674 (1912).

[6] *Metallurgical and Chemical Engineering*, vol. 12, No. 1, p. 54 (January, 1914).

[7] The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes, *Trans.*, vol. 39, 648 (1908).

[8] Bleioxyd und Kieselsaüre, *Metallurgie*, vol. 4, No. 19, p. 647 (Oct. 8, 1907).

[9] Über die Bildung von Bleisilikaten und ihre Rolle bei den neuen Bleigewinnungs prozessen, *Metallurgie*, vol. 5, No. 18, p. 535 (Sept. 22, 1908).

[10] Über den Blei-Rostreaktionsprozess, *Metallurgie*, vol. 4, No. 13, p. 455 (July 8, 1907).

[11] The Velocity of Reduction of the Oxides of Lead, Cadmium, and Bismuth by Carbon Monoxide, and the Existence of the Suboxide of these Metals, *Journal of the Chemical Society*, vol. 93, pp. 154 to 164 (1908).

[12] *Comptes Rendus*, vol. 128, No. 5, p. 309 (1899).

[13] S. E. Bretherton: The Treatment of Complex Ores by the Ammonia-Carbon Dioxide Process, *Trans.*, vol 49, 802 to 808 (1914).

[14] Irving A. Palmer: Smelting Lead Ores in the Blast Furnace, *Trans.*, vol. 49, 507 to 524 (1914); L. D. Anderson: *Idem*, 523.

[15] Hofman and Mostowitsch: The Reduction of Calcium Sulphate by Carbon Monoxide and Carbon, and the Oxidation of Calcium Sulphide, *Trans.*, vol. 41, 763 to 785 (1910).

[16] Hofman and Mostowitsch: The Behavior of Calcium Sulphate at Elevated Temperatures with Some Fluxes, *Trans.*, vol. 39, 628 to 653 (1908).

Determination of Dust Losses at the Copper Queen Reduction Works

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INTRODUCTORY

BEFORE the year 1909, no measurements of dust losses and flue gases had been made at the Copper Queen Reduction Works, at Douglas, Ariz. At that time the "unaccounted" loss of the smelter had reached a figure that drew attention to the possibility of large stack losses. The measurement of dust losses had not been studied much at that time, and estimates of dust losses, with a few exceptions, were the results of tests by what has been termed "chemical methods," that is, the aspiration and filtration through cotton, wool or some similar medium of a few liters of gas, the dust being weighed on a chemical balance, and the volume of gas computed from the volume of aspirator water. The possible error of this method is enormous, and the probable error much too large for reliable results. Since the object of these tests was to discover not only the extent of the dust losses, but also the possibility of profitably reducing them, it was decided that the method adopted must have the following characteristics:

1. The sample of gas taken for filtration must be representative of the whole volume of gas, that is to say, it must have the same dust content. This condition will be fulfilled in all cases if the gas sample is drawn off without change of rate of flow. Then it will contain the same proportion of entrained or suspended particles of dust as the original gas. In the case of flue gases which contain only fume, at a temperature above or near the sublimation point, this condition need not be adhered to since the fume will not behave as entrained particles, but more nearly as a true gas. Consequently the gas sample in this case only may be drawn off at any rate of flow.

2. The gas sample taken must be large enough (*a*) to yield sufficient dust to permit of accurate chemical and physical examination; (*b*) so that the ratio of the total volume of gas passing through the flue to the volume of gas sample taken shall not be too large.

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3. The methods of measurement of gas volumes should be such that an occasional independent check can be made.

Methods of Measurement of Dust Losses and Stack Gases

The dust loss determinations so far made at the Copper Queen smelter fall under two heads:

1. Determination of stack losses. Here gas velocities are relatively high and may be measured with accuracy by Pitot tubes.

2. Determinations of the loss from one department made by measuring the amount of dust passing a certain point in the flue, deducting the dust collected in the remainder of the flue. The gas velocity in most of these cases was too low to permit of measurement by Pitot tubes, so the static balance method of sampling had to be adopted.

Preliminary Measurements and Instruments Used

Velocity head is measured by Pitot tubes of the form recommended by W. C. Rowse,[1] and Ellison differential draft gages, using petrolic ether instead of the gage oil supplied. The former has the advantage of being less viscous and also lighter, but in the summer in Arizona it is necessary to shade the gage carefully as evaporation of the ether would otherwise be too rapid.

Atmospheric pressure is determined by a standard mercury barometer and flue temperatures by a Bristol (indicating) pyrometer or a Bristol recording thermometer. For the temperatures of the gas sample, a 600°F. mercury thermometer is used.

In order to convert velocity head from inches of water into feet of gas, it is necessary to know the specific gravity of the gas. This is computed from the analysis of the gas. It is not within the scope of this paper to give more than the briefest outline of the methods of gas analysis.

Sulphur trioxide, sulphur dioxide and carbon dioxide are determined together; SO_3 by the Hawley method;[2] SO_2 by absorption in a measured volume of a standard solution of Na_2CO_3 in the presence of excess H_2O_2; and CO_2 gravimetrically. In addition, oxygen and water vapor are determined, the remainder being taken as nitrogen.

It is found in practice that the specific gravity of all our gases, at points used for dust-loss determinations, closely approximates to 4 per cent. heavier than air. However, measurements are made before each set of tests.

Fig. 1 shows the arrangement of the apparatus used in the first series of tests. An 8-in. pipe was placed outside of the steel stack from the top to the base. The upper end of the pipe was extended to a point inside

[1] *Transactions of the American Society of Mechanical Engineers*, vol. 35, p. 643 (1913).

[2] *Engineering and Mining Journal*, vol. 94, No. 21, p. 987 (Nov. 23, 1912).

the stack, 7 ft. below the top. The method of varying the position of the suction intake is shown on the sketch. The first series of tests was made at five different points, but since no appreciable difference in dust loss could be found, subsequent tests have been made with the suction intake in position No. 1 only. The 8-in. pipe was connected to a No. 4 Sturtevant fan which was driven by a 5-hp. variable-speed motor. The fan discharged through an 8-in. diameter flexible connection into a 4-ft. diameter vertical drum of boiler plate. The outlet at the top of this was

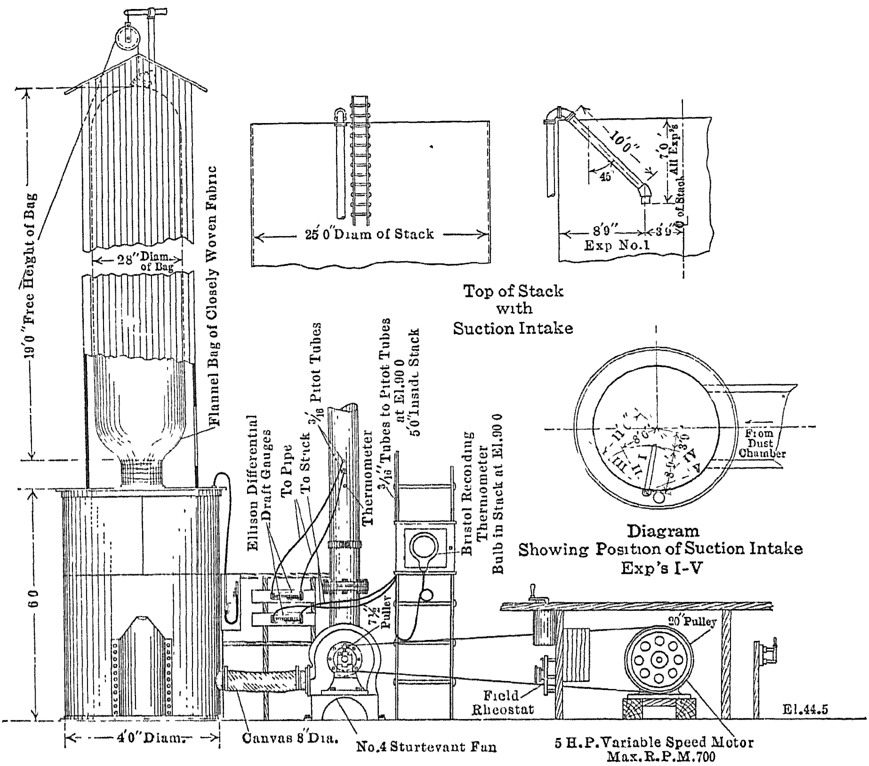


FIG. 1.—GENERAL ARRANGEMENT OF APPARATUS USED IN FIRST SERIES OF TESTS TO DETERMINE DUST LOSSES.

a 12-in. nipple, to which was wired a bag 28 in. in diameter by about 20 ft. long made of closely woven flannel. The top of this bag was closed by a cord, the end of which passed over a pulley. This afforded a means of shaking the dust out of the bag into the drum. The bag was housed in a wooden structure. A set of Pitot tubes was placed 5 ft. inside the stack, 90 ft. above the base of the stack, the temperature of the gas being measured at the same point by a Bristol recording thermometer. Another set of Pitot tubes and a mercury thermometer in the 8-in. pipe were used to measure the gas sample.

The basic formula $v = \sqrt{2gh}$ gives v the velocity of the gas in feet per second, if g the acceleration due to gravity is in feet per second per second, and h the velocity head is in feet of gas at the temperature and pressure of the gas. Now if P is the reading of the differential draft gage in inches of water giving difference between pressure in dynamic and static Pitot tubes; T, the temperature of the gas expressed in the absolute scale; Sw the specific gravity of water at the temperature of the gage; Sg the specific gravity of the gas at a temperature T, then

The velocity head $= \frac{Sw}{Sg} \times \frac{P}{12}$

Since the pressure of the gas is constant, its specific gravity varies inversely as the absolute temperature. Hence h is proportional to PT and $v = k_1\sqrt{PT}$, k_1 being a constant. Now if W is the weight of gas that passes a cross-sectional area of 1 sq. ft. in 1 sec. then

$$W = v \times Sg \times 1 = k_1\sqrt{PT} \times \frac{K_2}{T} = K\sqrt{\frac{P}{T}}$$

In order to obtain the same weight of gas per square foot per second in both stack and pipe (which insured having the same velocity of gas in stack and suction intake), it was only necessary to run the motor at such a speed that the quotient of P divided by T was the same for both stack and pipe readings.

Given the velocity and specific gravity of the gases, it is obvious that volumes and weights of gas can be easily computed. The volume of the gas sample, however, will be high if taken as the product of this velocity by the area of the sample pipe, owing to the fact that the center velocity is higher than the average. Readings should be taken along a diameter at the centers of annular rings of equal area and the average computed from them. This gives a factor for correcting center readings. This factor should be determined for several different velocities, as it depends on the velocity of the gas and the size and nature of the pipe. The early tests were not corrected in this way, and the dust losses are therefore lower than they should be. In subsequent tests, this correction has been made. After running the fan at the correct speed for a number of hours, the dust collected was cleaned up, weighed, sampled, and assayed, and the dust and metal losses computed.

The first tests (1909–1910) and the dust-settling experiments were made by the late E. T. Norlander, G. B. Lee being superintendent of the smelter at that time.

The method at present used, where high gas velocities are encountered, is the same in principle and is shown diagrammatically in Fig. 2. The measurements are made in the same way, but the gas sample goes through the filter under suction instead of under pressure, and the filter is made of asbestos fabric instead of flannel. A large fan is used on account of the

increased head required to pull the gas through the heavy asbestos fabric This bag is weighed before and after the run and the dust brushed out of it. The dust will contain some asbestos, but this may be removed by screening through a coarse sieve.

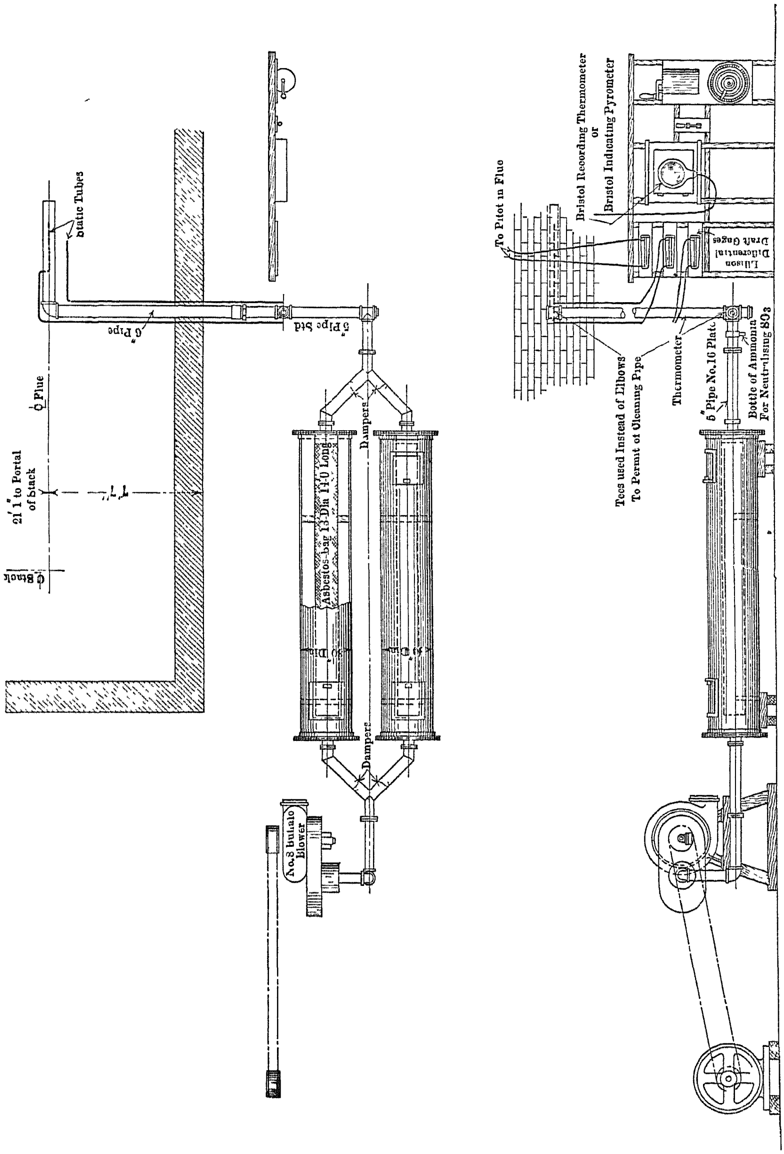


FIG. 2.—APPARATUS NOW USED FOR QUANTITATIVE DETERMINATION OF FLUE OR STACK GASES AND DUST LOSSES.

This method has several advantages: When filtering the gas under pressure any leaky place between the fan and the filter will result in the escape of gas carrying its dust with it; but when filtering under suction,

slight leaks do not matter since they only allow outside air to be admitted. Measurements of the gas are made in front of the filter, in the pipe, which can easily be made air-tight. So a correct measurement of the gas sample is obtained with the certainty that all of it goes through the filter.

The asbestos fabric has the advantage over flannel that it can be used for all smelter gases. Flannel was satisfactory for the blast-furnace and converter gases, but was destroyed almost immediately by the roaster gases, on account of their higher SO_3 content; for the first series of determinations of roaster loss, it is true, flannel was used, but it was necessary to put ammonia in the path of the gas and that resulted in the dust being contaminated with ammonium sulphate and sulphite. The net effect of this was to make our screen analyses of dust of no value. A filter of 200-mesh phosphor bronze screen was tried; it was not affected by the gases chemically and filtered efficiently, but it had no strength, and the expansion and contraction due to temperature changes disrupted it very quickly.

Method where Low Gas Velocities are Encountered

Fig. 3 is a plan of the Copper Queen Reduction Works. It will be seen from this that in order to determine the dust loss from any one department, recourse must be had to measurements in one or other of the flues. The converter and roaster losses have been so determined. In both cases the point of sampling was fixed by the arrangement of the flues. Both places left much to be desired, but it was decided that by careful work and by obtaining as exact knowledge as possible of flue conditions, a sufficiently accurate result would be obtained. The velocity of the gases in these flues is such as to give a just measurable velocity head. Accordingly, Pitot tubes were used in conjunction with a sensitive Ellison gage, as one method of gas determination, but more reliance was placed in the measurement of flue gas velocity by the static balance method. Two static tubes are placed, one inside the sample pipe and the other in communication with the flue only, and connected to a differential draft gage. By adjusting the speed of the motor so that zero reading is obtained on this gage, the gas flows into the suction intake at the same velocity as in the flue, around or near the sample pipe. Outside the flue the gas travels through a pipe of smaller diameter than the suction intake. This results in a magnification of the velocity in the ratio of the areas of the pipes, and permits of accurate velocity measurements.

If v is the velocity of the gas in this smaller pipe, T is its temperature, T_1 is the temperature of the flue gases, both temperatures being in the absolute scale, and if D is the diameter of the suction intake, d the diameter of the smaller pipe, then the velocity of the flue gases around the

sample pipe

$$= v \times \frac{d^2}{D^2} \times \frac{T_1}{T}$$

In using this method we arrange the piping so as to measure a velocity

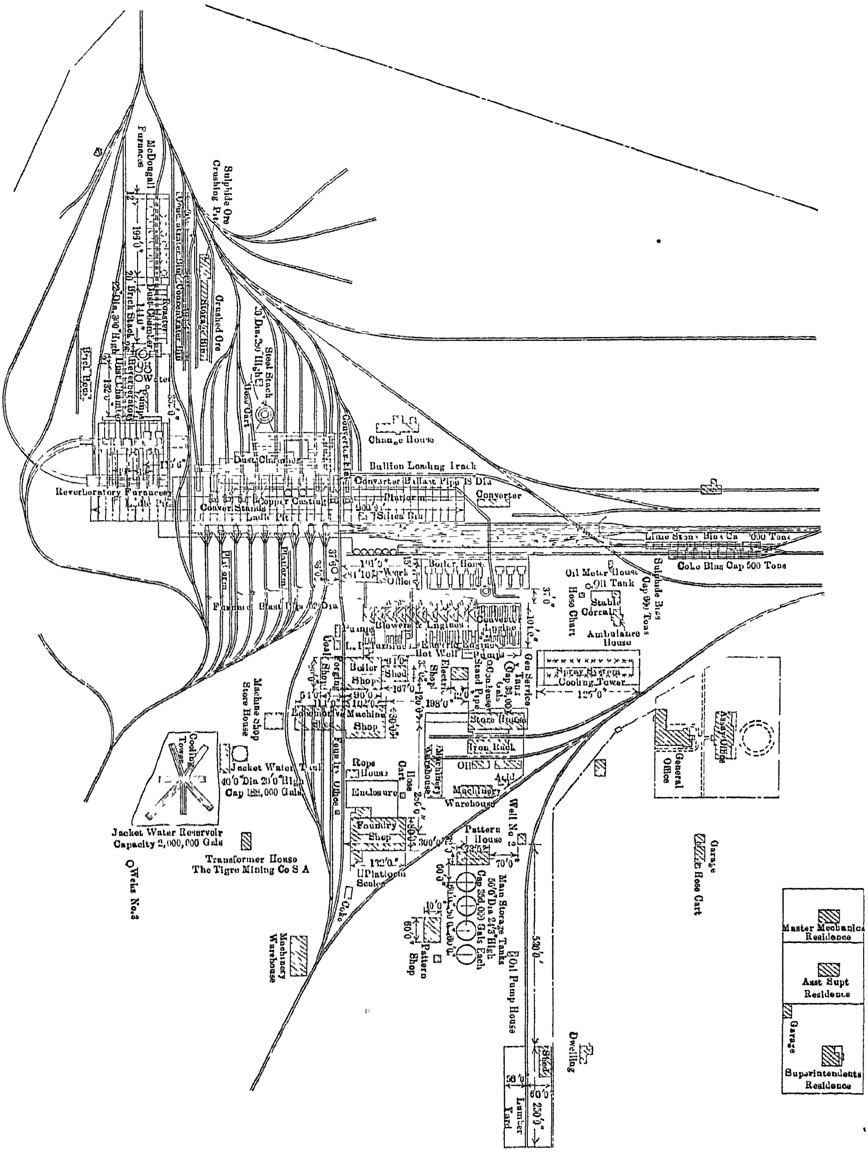


Fig. 3.—Plan of the Reduction Works of Copper Queen Consolidated Mining Co., Douglas, Ariz.

of between 30 and 40 ft. per second.

The volume of the gas sample, indicated by the Pitot tube in the small

pipe, gives a measure of the velocity of the gas at the suction intake; it is only in rare cases that the rate of flow of the gas will be uniform over the whole cross-sectional area of the flue. In order to compute the volume (V) of gas flowing through the flue, it is necessary to obtain a factor to convert this velocity into volume. The free cross-sectional area of the flue is first determined by taking soundings of the dust line at a number of points across the flue. These can then be plotted and the area determined with a planimeter.

If v is the velocity of the gas at the sample point, A the free cross-sectional area of the flue, then

$$V = KAv$$

K the constant being the ratio of the average velocity of the gases over the whole flue to the velocity at the suction intake. Two methods have been used to determine K. The most direct method is to make a careful exploration of the flue with two sets of Pitot tubes, one being kept at the position of the suction intake, and the other moved to a series of points across the flue. The cross-section of the flue is divided into a number of equal areas and these readings taken at the centers of such areas. From these readings, the volume of flue gases is computed and the factor K obtained and used in the dust-loss tests.

The other method of determining K can be used where gas velocities are too low for Pitot tube readings, or as a check in Pitot work. Velocity readings with Pitot tubes are taken at some point in the system where the cross-sectional area of the conduit is a minimum, a sufficient number being taken to give a fair average. Simultaneous gas samples are then taken at this point and in the flue for which K is to be determined and these gas samples are analyzed for one or more constituents such as SO_2, CO_2, or O. Since the volume of gas passing the first point is known, and the simultaneous gas analyses give the percentage of excess air admitted into the system between two points, the volume at the second point can be obtained from these and a comparison of the gas temperatures at the two points.

The exploration of the roaster flues showed very great variation in gas velocity across the flue. From this we inferred that the filtration of a gas sample from any one point would give us a dust loss that was too high or too low according to whether the suction intake was in a zone of high or low velocity. Accordingly, we decided to make small-scale tests to give us the comparative dust content of the gases in different parts of the flue. These were referred to the suction intake of the large-scale tests. The apparatus used for this was developed by the engineers of the Western Precipitation Co., and has been described by W. N. Drew.[3]

[3] *Journal of American Society of Mechanical Engineers*, vol. 38, No. 12, p. 676 December, 1915).

Briefly it consists of a small suction intake with two statics as in the large-scale work. We used ¼-in. and ⅜-in. diameter pipe. The gas was filtered through a weighed fat-extraction thimble. The gas was drawn out by aspirator, and the rate of flow adjusted to zero on a differential draft gage connecting the two statics. Velocities were computed from the volume of water drawn from the aspirator, corrected to tempera-

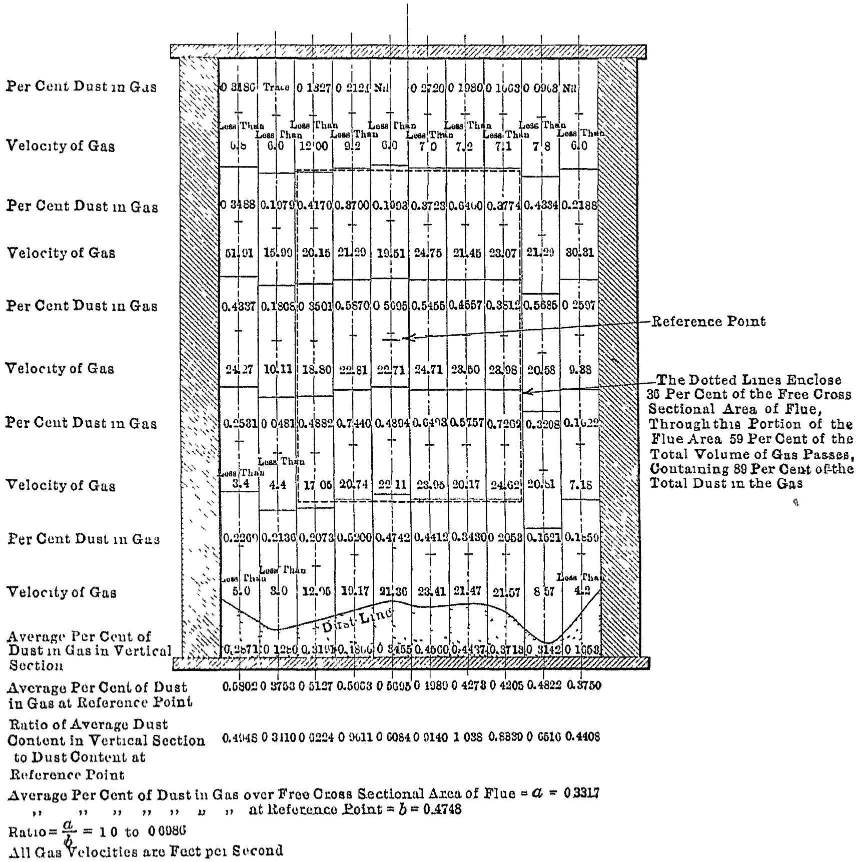


FIG. 4.—VARIATION OF DUST CONTENT OF GASES IN ROASTER FLUE.

ture and pressure conditions in the flue. About 12 cu. ft. of gas constituted a sample, and the dust from it weighed from 0.1 to about 4.0 g.

The width of the flue was divided into 10 equal sections and readings taken at the centers of each section. One section and the check readings at the suction intake were determined per day, and the average dust content of the gases per section referred to the suction intake. The average of the 10 ratios so obtained was used to correct the dust loss computed from the large-scale determinations at the fixed suction intake.

This correction has been applied only in the case of the roaster flue, other cases showing only minor variations in gas velocities.

Fig. 4 shows the variation in dust content of the gases in the roaster flue.

Determinations of Blast-Furnace and Converter Stack Loss

In 1909, when these tests were started, the smelting equipment consisted of five 42 by 240-in. and five 42 by 216-in. blast-furnaces and eight stands of converters, 8 ft. by 11-in. barrel type, acid-lined. Table 1 shows the dust losses at that time, with seven, eight and nine furnaces in operation. The precious-metal losses shown by these tests are not comparable with subsequent ones, owing to the fact that the blast furnace charge at this time contained some high-grade custom ore, which materially increased the precious-metal loss. Fig. 5 shows graphically the daily variations in the volume and weight and dust content of gases in the experiments.

Following these tests an experimental settling chamber of 23 sq. ft. cross-sectional area by 50 ft. (afterward 100 ft.) long was erected to make tests of dust settling at different rates of flow of gas. The results of these tests are contained in an article by G. B. Lee,[4] then Superintendent of the smelter. Mr. Lee in summarizing says, "We believe as a result of these experiments, that flues or chambers to settle dust need not be long. In fact, 125 ft. would appear to be enough if the velocity of gases does not exceed 150 ft. per minute, and that this speed may be materially increased if wires or screens are placed across the direction of flow."

After these experiments, the dust chamber was enlarged so that the gases from two furnaces nearly conform to the condition Mr. Lee lays down. At the same time a balloon flue 11 ft. in diameter at one end and 13 ft. in diameter at the other was built to handle the converter gases which previously discharged directly into the dust chamber.

Table 2 has a sketch plan of the dust chamber showing the change. Tests of the stack losses were made in 1911 following this change. Comparing the results obtained with previous tests made under the same operating conditions, there is a decrease in dust loss of 103,800 lb. per day (from 164,680 to 60,860), and in the copper loss of 10,800 lb. per day (16,142 to 5,355). That is a decrease of 63 per cent. of dust loss and of 67 per cent. of the copper loss.

It will be seen from the sketch plan of the extended dust chamber that two blast furnaces only discharge into the extended portion and that the header flue connecting this extension with the stack receives also all the converter gases. Two series of tests were made of the loss from the end of this header flue, the first with both furnaces and the converters

[4] *Engineering and Mining Journal*, vol. 90, No. 10, p. 504 (Sept. 10, 1910).

in operation, and the second with the converters only in operation, but enough air being admitted through the furnace uptakes to make the gas velocity in the header flue as nearly normal as possible. The results of

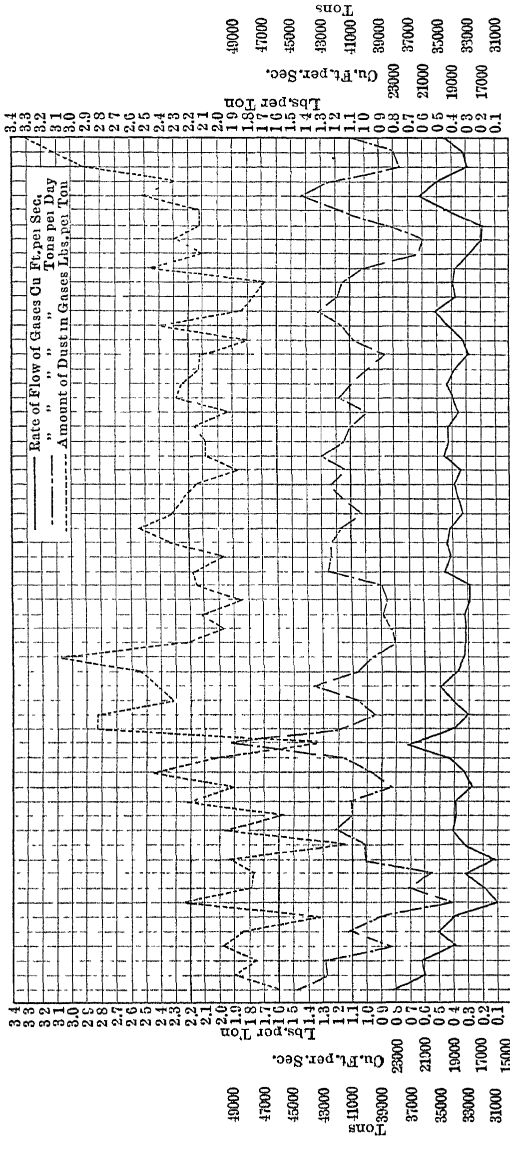


FIG. 5.—DIAGRAM SHOWING RATE OF FLOW OF GASES AND AMOUNT OF DUST IN GASES.

these tests are given in Table 2. Recently we made determinations of the converter loss alone, in the same place, but the results are not comparable, as seven 12-ft. upright, basic-lined converters have replaced the eight small acid-lined converters.

In 1913, after the reverberatory and roaster plant had been blown in, the fine ore was no longer charged into the blast furnaces. Another series of tests was made under these conditions which shows a further decrease, the dust loss being 40,000 as against 60,000, and the copper loss 2,589 as against 5,355. It should be noted, however, that one less blast furnace

TABLE 1.—*Quantitative Determination of Stack Gases and Dust Losses*

Experiment No.	Number of Runs	Duration of Runs, Hr	Blast Furnaces in Operation	Converters in Operation, No	Velocity-Head, Inches, Water		Temp., °F.		Velocity, Ft per Sec		Volume of Stack-Gases, Cu. Ft per Min	Weight of Gas, Pounds			
												Per Sq Ft per Sec.		Total During Run	
					Fan	Stack	Fan	Stack	Fan	Stack		Fan	Stack	Fan	Stack
1–5	60	7@8	7	6	0.1727	0.2175	110	258	30.31	38.27	1,127,000	1.9053	1.9069	17,953	25,260,000
6	5	4½	8	7	0.176	0.247	120	349	30.81	43.07	1,268,000	1.9162	1.9134	11,078	15,524,000
14	7	6	9	6	0.275	0.374	114	320	38.35	52.15	1,533,000	2.3690	2.4180	18,063	25,353,000

Weight of Dust, Pounds				Amount of Dust in Gases, Per Cent	Analysis of Dust												Values Escaping in 24 Hr.		
During Run		In 24 Hr.																	
Fan	Stack	Fan	Stack		Au, oz.	Ag, oz	Cu, %	Pb, %	Insol, %	SiO_2, %	Fe, %	Mn, %	CaO, %	Al_2O_3, %	S, %	Zn, %	Gold, oz	Silver, oz	Copper, Lb
19.57	27,698	62.64	88,633	0.109	0.02	5.86	8.78	8.6	22.4	17.3	12.4	0.4	2.5	9.4	11.9	5.3	1.01	262.4	7,801
17.81	25,048	92.95	130,523	0.161	0.023	6.63	9.00	5.9	25.6	20.3	14.7	0.5	1.5	8.5	11.1	4.2	1.49	435.9	11,747
30.98	41,170	125.60	164,680	0.162	0.02	3.42	9.67	3.3	25.8	19.5	14.6	0.5	2.7	10.8	11.4	4.4	1.61	280.3	16,142

was in operation. Table 2 is an abstract of all these determinations of the blast-furnace and converter stack loss and shows the effects of the extension of the dust chamber with the removal of the converter gases from the main body of the chamber and also the removal of fine ore from the blast-furnace charge.

Converter Losses

We made two series of determinations, one of the converter-dust loss, and one of the amount of dust passing the end of the converter balloon flue. The testing department of the Anaconda Reduction Works had discovered that by leaching their converter fume with dilute sulphuric acid they could obtain a concentrate high enough in lead to justify recovery of that metal, the copper and zinc being recovered by precipitation on iron, or by electrolytic precipitation. From the converter flue we recover dust high in copper (50 to 60 per cent. Cu), but containing only small quantities of Pb and Zn. Passing that flue, the fume and dust together form a product that contains from 5 to 20 per cent. Cu, 17 to 29 per cent. Pb, and 6 to 12 per cent. Zn. We found that an acid leach is not necessary for our material, water giving nearly as good an extraction, and by using a water leach, followed by a leach with a solution of ferrous and ferric sulphates, an extraction of 74 per cent. of the copper, and 97 per cent. of the zinc was obtained with the formation of a concentrate containing 55 per cent. Pb. The ferrous and ferric sulphate solution would be obtained from the precipitation of the copper liquors on iron.

The quantities involved, however, are small and the project is still under consideration.

Roaster Dust-Loss Determinations

The roaster and reverberatory plant originally comprised six 18-ft. diameter, six-hearth McDougall roasters, and two 19 by 91-ft. 6-in. reverberatory furnaces, the flue gases from each of which are conducted through two 520-hp. Erie City waste-heat boilers. The plant has since been enlarged to 16 roasters and three reverberatories, of which 14 or 15 roasters and two reverberatories are normally in operation.

Separate dust chambers are provided for roasters and reverberatories. These dust chambers differ only in length, the roaster chamber being 12 ft. longer than the reverberatory; they are 144 ft. and 132 ft. long respectively. The cross-section of these chambers is shown in Fig. 6; its area is 1,294 sq. ft., but the free cross-sectional area with the usual amount of dust in the bottom is about 1,235 sq. ft. Wires are hung in both chambers in all but the first and last 12 ft., so that the roaster chamber has 120 ft. of its length, the reverberatory 108 ft., in which wires are suspended. A more complete description of the Copper Queen smelting works is given by Richard H. Vail.[5]

The reverberatory dust losses have not been determined. The dust settles mostly in the waste-heat boilers, very little settling in the chamber.

[5] *Engineering and Mining Journal*, vol. 99, No. 1, p. 1 (Jan. 1, 1915).

The dust loss is probably nominal and it has never been considered necessary to determine it.

The roaster losses have been determined under the following different conditions of operation:

TABLE 2.—*Abstract of Determinations of Stack Gases and Dust Losses. Blast Furnace Stack*

Number	Date	Conditions and Purpose of Experiment	Number of Runs	Average Duration of Runs, Hours	Temperature of Stack-Gases, ° F.	Number of Blast Furnaces in Operation	Number of Converters in Operation	Blast Pressure: Blast Furnace, Oz.	Blast Pressure: Converters, Lb.	Volume of Gas Passing up Stack, Cu Ft per Min.	Weight of Gas, Lb: Per Square Foot Per Second: Fan	Weight of Gas, Lb: Per Square Foot Per Second: Stack	Weight of Gas, Lb: During Run: Fan	Weight of Gas, Lb: During Run: Stack	Weight of Dust, Lb: During Run: Stack
1	May 1909	Determination of dust losses with original arrangement of dust chamber.	60	7–8	258	7	6	....	.....	1,127,000	1.9053	1 9069	17,953	25,260,000	27,698
2	Sept 1909	Same as above.....	5 7	4½ 6	349 320	8 9	7 6	....	....	1,268,000 1,533,000	1.9162 2 3960	1 9134 2 4180	11,078 18,063	15,524,000 25,353,000	25,048 41,170
3	June, 1911	Dust chamber extended and converter flue erected Determination of effect on dust losses	15	6½	344	9	7	25	9 6	1,239,000	1 872	1 877	15,257	21,531,000	16,483
4	Nov. 1911	Sample taken from brick header flue Determination of loss from furnaces No 1 and No. 2 and all converters	16	7	Header 294	2	6	28	10 5	Header 302,050	0.759	Header 0.757	6,709	Header 6,150,900	Header 5,120
5	Feb 1912	Sample as above Furnaces No. 1 and No. 2 Down. Determination of converter loss alone	14	6½	198	0	6	0	9 75	282,700	0 793	0 796	6,477	6,008,900	1,679
6	Sept. 1913	Same as experiment 3 but no fine material [Concentrates or flue dust] Charged into blast furnaces	12	6	Stack 318	8	7	28	12 25	Stack 997,200	1.571	Stack 1 568	11,675	Stack 16,409,000	Stack ....

For the first five series of tests, averages are given, for series No. 6 an average run (made 9-13-13) is reported.

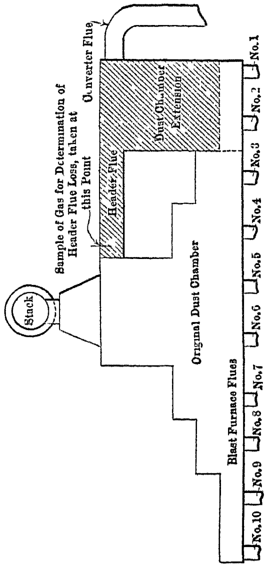
	Roasters in Operation	Draft	Feed
(1)	8	Low	Fine
(2)	15	Medium	Mixed
(3)	15	High	Mixed
(4)	15	Low	Mixed
(5)	14	Low	Coarse

Tests (1) were made with a 10-in. suction intake and a 5-in. pipe outside the flue; tests (2) to (5) with a 6-in. suction intake and 5-in. outside.

TABLE 2.—*Abstract of Determinations of Stack Gases and Dust Losses. Blast Furnace Stack.—(Continued)*

Weight of Dust, Lb. In 24 Hr Stack	Amount of Dust in Gases, Per Cent.	Tons of Ore Charged in 24 Hr.	"Ore" lost as "Dust" Per Cent. of Total Charged	Analysis of Dust: Oz a Ton: Gold	Silver	Per Cent: Cu, Copper	Pb, Lead	Insol	SiO_2, Silica	Fe, Iron	Mn, Manganese	CaO, Lime	Al_2O_3, Alumina	S, Sulphur	Zn, Zinc	Values Lost in 24 Hr.: Gold, Oz	Silver, Oz	Copper, Lb	Lead, Lb.	Zinc, Lb.
88,633	0 109	2,362	2 76	0.02	5 86	8 78	8.6	22 4	17 3	12 4	0 4	2 5	9 4	11.9	5 3	1 01	262	7,801	7,622	4,697
130,523	0 161	3,009	2.73	0.023	6 63	9 00	5 9	25 6	20 3	14 7	0 5	1.5	8 5	11.1	4 2	1 49	436	11,747	7,700	5,481
164,680	0.162	...	......	0.02	3.42	9.67	3 3	25 8	19.5	14 6	0.6	2.7	10.8	11 4	4 4	1.61	280	16,142	5,434	7,245
60,861	0.076	2,962	1.03	0.01	4 8	7.90	....	20.3	14.8	13.5	0 5	1.8	8 1	12.7	5 6	0.30	145	5,355	. .	3,408
Header 17,556	0 083	. . . .	. . . . .	0 06	10 6	9 35	13 7	.	11 4	124	.	. . .	5 5	12.0	5 89	0.52	93 1	1,644	2,416	937
6,200	0.027	......	.....	0.01	7.8	5 97	21 4			..	.		.	..	9.78	0 56	24 4	374	1,338	625
Stack 40,910	0.0615	2,985	0.671	Nil	3 2	6 33	5 7	18 0	13 4	11 0	0 6	1 5	7 0	12 0	7 0	. .	64 2	2,589	2,286	2,808

For the first five series of tests, averages are given; for series No 6 an average run (made 9-13-13) is reported.



The averages of all these tests are given in Table 3, and Fig. 7 is a graphical representation of the variations in tonnages roasted, volumes of the gases, dust and copper losses, the last two being plotted to much larger scales than the tonnages roasted. The dust-chamber walls are

constructed of hollow tile laid in cement mortar. The conversion of the calcium carbonate in the mortar into calcium sulphate by the sulphur trioxide in the gases caused a volume change that shattered the tile. This

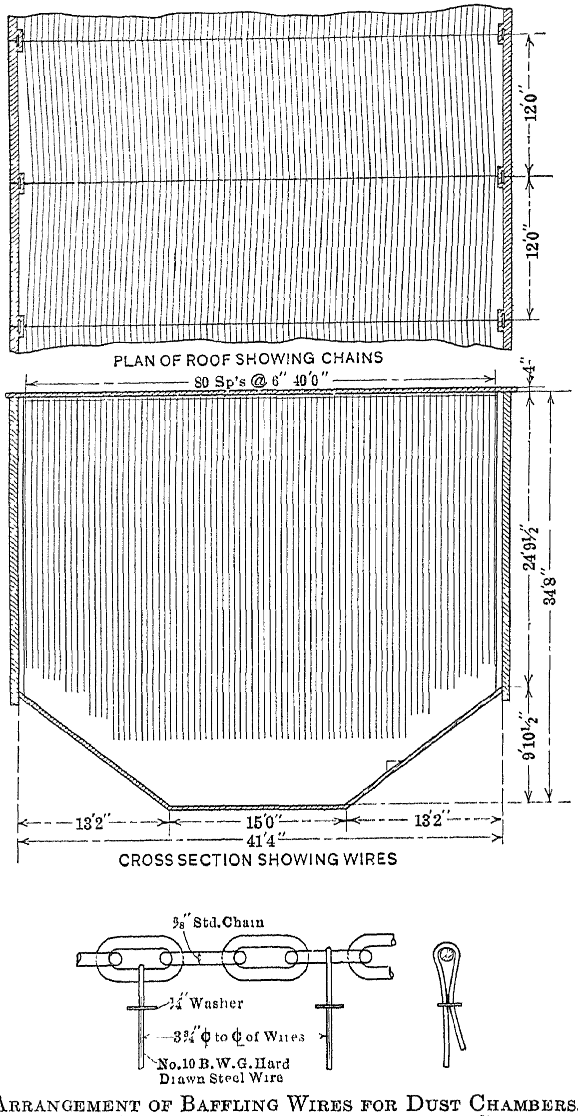


Fig. 6.—Arrangement of Baffling Wires for Dust Chambers. Detail Showing Method of Attaching Wires to Chains.

resulted in a large leakage of air through the walls of the chamber at the time of tests (2) to (5). In fact, this "false air," admitted through the dust chamber walls, constitutes approximately one-third of the total volume of flue gases. The walls are at present being repaired,

the damaged sections being replaced by new tile laid with a fire- and acid-proof mortar. When the repairing is completed, another set of tests will be made. We anticipate that there will be a considerable reduc-

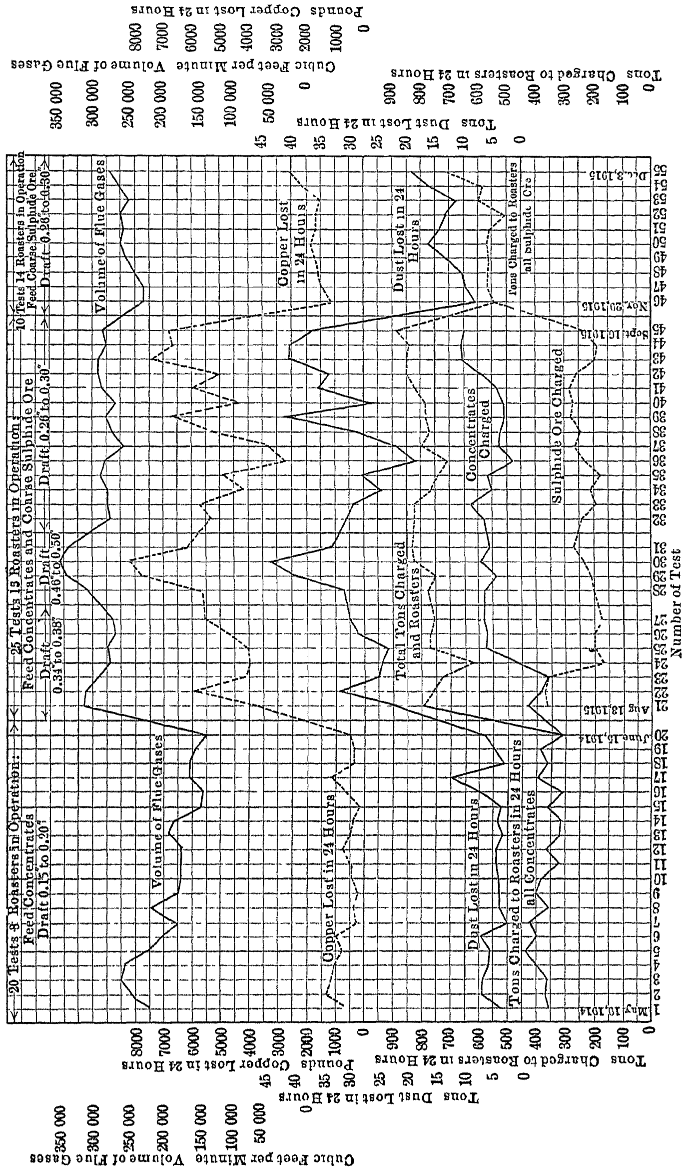


FIG. 7.—QUANTITATIVE DETERMINATIONS OF ROASTER GASES AND DUST LOSSES UNDER DIFFERENT OPERATING CONDITIONS.

tion in the dust loss. For should the stoppage of this air leakage not result in any decrease in the percentage of dust in the gases, the dust loss would still be less in the proportion of the diminution in volume. It

is, however, highly improbable that the decreased volume, moving through the chamber at a proportionately reduced velocity, would carry as high a percentage of dust at the outlet.

The roaster dust loss seems to depend on the tonnage treated by the roasters, the roaster draft, and the size of the particles fed. It is found

TABLE 3.—*Quantitative Determinations of Roaster Gases and Dust Losses*

Average of Experiments	Dates	Duration of Tests, Hr.	No. of Roasters in Operation during Tests	Gas Sample: Averages: Temperature of Gas, °F.	Gas Sample: Averages: Velocity of Gas in 5-In. Pipe, Ft. per Sec.	Gas Sample: Averages: Volume of Gas, Cu Ft. per Min.: At Gas Condition	Gas Sample: Averages: Volume of Gas, Cu Ft. per Min.: Standard Conditions	Weight of Gas, Sample Taken during Test, Lb	Weight of Dust Obtained, Lb	Per Cent. of Dust in Gas	Flue Gases: Averages: Temperature of Gas, °F.	Flue Gases: Averages: Velocity of Gas in Flue, Ft. per Sec	Flue Gases: Volume of Gas, Cu Ft per Min.: At Gas Condition	Flue Gases: Volume of Gas, Cu Ft per Min.: Standard Conditions	Weight of Gas, Lb: Per Min.	Weight of Gas, Lb: During Run	Weight of Gas, Lb: In 24 Hr.
1–20	5-19-14 to 6-15-14	5–6	8	220	46 92	472.9	296.4	8451	5.68	0.0812	298	13.15	190,780	107,300	8,800	3,171,000	12,680,000
21–27	8-13-15 to 8-18-15	3–5	15	280	30 41	248.5	143.9	3191	14.17	0 4409	367	28 51	287,500	148,800	12,340	3,313,000	17,770,000
28–31	8-21-15 to 8-27-15	5	15	276	34.20	279.4	162.9	4031	21 12	0 5209	359	26.34	330,100	173,000	14,320	4,296,000	20,615,000
32–45	8-28-15 to 9-16-15	4–5	15	262	26 72	216 9	129.5	3005	15 59	0.5265	366	22.29	280,100	146,000	12,010	3,384,000	17,350,000
46–55	11-29-15 to 12-9-15	5	14	222	21 91	179.1	113.4	2742	6.92	0 2495	346	17 98	251,000	134,500	11,055	3,254,000	15,920,000

Tons of Charge Roasted in 24 Hr.	Dust Loss: Lb. in 24 Hr.	Dust Loss: Per Cent. of Roaster Charge	Dust Loss: Grains of Dust per Cu. Ft of Gas	Analysis of Dust: Oz per Ton: Au	Oz per Ton: Ag	Percentages: Cu	Pb	Insol	SiO_2	Fe	CaO	Al_2O_3	S	Zn	Metal Loss in 24 Hr: Gold, Oz.	Silver, Oz	Copper, Lb	Draft at End of Inlet-Flue to Roaster Dust-Chamber, Inches of Water	Roaster Feed
361	10,160	1 414	0.259	Trace	3 44	5.95	0 4	33 3	22 3	8 6	0 7	8 0	16 3	0 7	. .	17 6	605	0 15 to 0.20	1
752	52,704	3 508	0.894	0.03	3 85	8.68	0 2	31.8	22 8	16 3	1 3	6 4	15 2	0 8	0.75	101.7	4,589	0 34 to 0 38	2
795	71,940	4.623	1 077	0.03	4 02	9 44	0.1	33 5	24 0	14.9	1 2	7.05	15 5	0 9	1.04	147.5	6,941	0 46 to 0 50	2
806	62,000	3 831	1 072	0.02	3.61	8 51	0 1	32 1	24 3	13.5	1 0	6 8	15 5	0 5	0 66	111 9	5,308	0 26 to 0 30	2
576	25,950	2.243	0.478	0 04	1.61	6 73	Nil	31.0	20.7	17.4	1.8	7.8	12 2	1 5	0.50	20 5	1,726	0 26 to 0 30	3

[1] Concentrates. [2] Concentrates and sulphide ore. [3] Sulphide ore.

in practice that the roaster draft has a great effect on the capacity of the furnaces, but our tests also show that increase of draft, or increase of tonnage treated, result in prohibitive dust losses, so that now the roaster draft is kept at the absolute minimum required to carry away the gases.

The last set of experiments (5) does not represent normal working

The activities of General Villa in Sonora cut off the supply of concentrates from the Moctezuma Copper Co., and gave us the opportunity to determine the effect of the size of the feed on the roaster loss.

Conclusions

The value of this class of work cannot be overestimated. The amounts involved are usually many times the outlay required for carrying on the work, and the systematic study of dust loss will always prove a good investment. The results of tests of blast-furnace and converter

TABLE 3.—*Quantitative Determinations of Roaster Gases and Dust Losses (Continued)*

	Typical Analyses										
	Oz. per Ton		Percentages								
	Au	Ag	Cu	Zn	Insol.	SiO_2	Fe	CaO	Al_2O_3	S	Mn
Concentrates	0.007	3.50	13 90	..	16 0	11.7	31.3	0 5	3 0	35 0	0.3
Sulphide ore	0.035	0.57	4.16	.....	26 8	21.0	30 3	0.8	3.9	31.6	0.3
Calcined concentrates	0.03	4.33	16 18	1.1	......	12 8	36.3	0 6	3.4	11.8	
Calcined sulphide ore	0.07	1.53	6.36	...	.. .	19.0	38 0	1 9	3 4	8.0	0 1
Chamber dust	0.03	2.97	11 40	1 2	29 2	21.1	25 7	1 6	5 6	20 7	0 6

	Screen Analyses											
	Per Cent. Retained on Screen of											Per Cent. through Last Screen
	2 Mesh	3 Mesh	4 Mesh	8 Mesh	15 Mesh	40 Mesh	80 Mesh	120 Mesh	150 Mesh	180 Mesh	200 Mesh	
Concentrates	.....	8.0	6.9	19 7	.....	25 4	11.4	5 2	3 1	2 2	3 4	10.3 through 200 mesh
Sulphide ore	19.4	18 2	6 0	15.8	7 5	22.7	4 9	1 6	1 0	.. .		2.9 through 150 mesh
Dust loss expts. 1–20	.. ..	. ...	.....	.....	.....	..	.		. .	....	2 0	98.0 through 200 mesh
Dust loss expts. 21–45	.....	. ...	. ...	......	.	....	0 2	0 4	0 7	10 3	.....	88.4 through 160 mesh
Dust loss expts. 46–55	.....	.....	.....	.....	...	0.5	1.1	0.4	0.4	0.5	0.6	96 4 through 200 mesh

N. B. Expts. 1–20: Diameter of suction intake 10 in. reduced to 5 in. outside of flue. Expts. 21–55. Diameter of suction intake 6 in. reduced to 5 in. outside of flue.

losses show what can be accomplished, once the order of these losses is known.

To reduce the high dust losses from the roasters, changes in the flue system are being made. The reverberatory dust chamber which at present collects but little dust will be used as an additional chamber for the roaster gases. The present plan is to build two new flues, one to carry the roaster gases from the roaster dust chamber to the inlet end of the reverberatory chamber through which they will pass on their way to the stacks. The other flue will carry the reverberatory gases from the header at the back of the boilers directly to the stack. By these changes, the roaster gases will have 132 ft. more of travel, at low velocity, in which to deposit dust.

Although the advantages of this work are fairly obvious, the limitations should not be lost sight of. Estimates of dust losses by these or any other methods at present in use are subject to considerable errors. The average of a number of tests is a close approximation to the truth. One set of tests gives us a figure that applies to the particular operating conditions of that time only and no estimate can safely be made from it of dust loss under different operating conditions. In using the results of these tests, we allow for a probable error of 10 per cent. Changes will probably improve our methods from time to time, but, in measuring the dust content of a flow of gas in a conduit, variations in flow and composition of gas, under ordinary operating conditions, are so great that we shall always have to depend on the accuracy of the average of a number of tests rather than on individual tests. In making these tests, we always keep a log of anything that can possibly have any bearing; individual tests will sometimes show big variations from the average, and with all the facts known, it is usually possible to explain such variations or discover some cause of error. Tests are never thrown out, however, except for cause—we do not make the practice of discarding all that vary more than a certain percentage from the average.

In conclusion, we know the order of accuracy of this work, and believe all sources of error that can be allowed for or taken into consideration, are allowed for, and that with judgment its results are most useful in checking and cutting down smelter losses. We welcome constructive criticism.

I wish to express my thanks to F. Rutherford, Superintendent of the Copper Queen Reduction Works, for permission to publish this account and the detailed information of dust losses that accompanies it.

Discussion

The Chairman (Walter Douglas, New York, N. Y.).—Perhaps there is no problem that causes the modern metallurgist more worry

than the question of unaccounted for loss. He has not the advantage of the metallurgist of the early days here, who could sleep peaceably with the settled conviction that there were certain copper mineral constituents of the ore which were volatile, and there was therefore nothing to worry about. The slag contents and volatilization accounted for the balance of the copper that went in the top of the furnace after the weighing of the bullion. There have been great strides in recent years toward reducing this loss. Perhaps the Anaconda company is the one we have to thank for the original pioneer work along the lines which we are all today following. We would be glad to hear from some gentlemen present with reference to their experience with this question of dust losses.

E. P. MATHEWSON, Anaconda, Mont.—You mentioned the Anaconda company, Mr. Chairman. I would like to say something about what we have been doing there, and the troubles we have been having. We have a tremendous volume of gas to handle. We adopted the system of large chambers, lessening the velocity of gases, and incidentally cooling the gases. Unfortunately in one sense and fortunately in another, we were compelled to increase the amount of gas handled in the flue system, and the results were then not so good. Still, we get a very good recovery. Our system of taking a sample is similar to the one described by the writer of the paper. We have tried all kinds of apparatus for getting a fair sample. We divided the area of the flue into imaginary squares, and have taken samples for a certain period of time from each square; and have used the asbestos bags. At present, there is a large unit of the Cottrell apparatus almost ready to connect up on the roaster flue to catch the dust from flotation concentrates. The worst things smelters have to handle at the present day, they are coming in the Southwest as well as the Northwest, are of material much of which is as small as 500 mesh, and the ordinary methods of catching dust will not apply to flotation concentrates. The Cottrell process is going to be tried out at our plant for this material. We have devised special forms, with a view to preventing the formation of dust from this material, and we are pleased with the results so far obtained. There is another point in connection with this subject that I want to put before the members here, so they will be thinking about it. You know there has been a great deal of litigation between the farmers and the smelting companies on account of the damage some smelter smoke causes. In the early days at Anaconda we were turning out a great deal of dust from our small flues and short stacks. That we overcame and we stopped the damage to such an extent that the courts sustained us throughout, and proved that our improvements were really improvements, and we had ceased to materially damage the farmers in our vicinity. The one thing that our good friends of the Bureau of Mines are apt to overlook in checking up smelters is the fact that smelters

must be run on a commercial basis. Metallurgy is the art of getting money from ore; and if you cannot get money from the ore, it is not metallurgy. It is a loss. It is not business, either. We have to look out for that when we figure on saving the material in the gases. There is a great deal of good material in the gases that go from smelters, and there is a great deal of material that is not good. There are certain places, as in the center of Germany and England, where it would pay to take everything in the gas and give it a chemical analysis, separating each element and putting it on the market. The best thing to do in handling smoke from smelters is to follow nature's laws as nearly as possible. If you take the smelter smoke and drench it with water, you get a mixture that cannot be deposited in the streams, and there is really nothing than can be done with it. It is worse than if you let the smoke go out, without attempting to purify it. If you cool the gases to a point where it is safe to put them through a bag house, you get sulphur dioxide cooled to a point where it does more harm than if you had let it alone. The thing to do is to find a happy medium, and get the gases escaping at such a temperature that they will go high in the air and diffuse. Gases do not diffuse as quickly as many would have us believe.

S. J. Jennings, New York, N. Y.—There is one point in this unaccounted for loss that has not been mentioned. The ore when received at the smelter is weighed, and in most cases thereafter is dumped on to a bed, sometimes from a greater height than others, but always from some height. When you have the dry condition that obtains in the Southwest with occasional high wind, you are going to have a dust loss before it gets in the furnace. My observations this morning have made me believe that this dust loss is material. I was interested in what Mr. Mathewson had to say. We have in Shasta County, a smelter which treats 800 or 900 tons of sulphide ore a day. It is putting out somewhere in the neighborhood of 350 tons of sulphur daily. All the smoke is filtered so that the gas which is going out of the stacks of the bag house is invisible. The gas that comes from the settlers and converter is likewise led into the bag house, so that there is no visible smoke in the smelter. Yet, some 15 miles away from that smelter, farmers have the imagination to claim that damage is being done by sulphur dioxide, their claim being that sulphur dioxide forms pistons in the valley of the Sacramento River; when the wind blows with the river current the pistons of concentrated sulphur dioxide travel at least 15 miles and cause discoloration of vegetation. The smelter, on the other hand, claims that there is no material damage done by the sulphur dioxide—that other things cause the discoloration—and the smelter thinks it has proved its case. The material that results from the filtering of the smoke is a most complex fume, containing a great many of the rarer elements—arsenic, bismuth, platinum, copper,

silver, zinc and some gold. By means of the electrolytic treatment of zinc, we have found that it is commercially profitable to treat this fume which has accumulated for some 4 or 5 years. The thing to do is to accumulate all material containing values and after a while you will find a process by which a profit can be made. That seemed to be our experience in Shasta County.

C. E. Arnold, Miami, Ariz.—I should like to make an inquiry as to the point raised by Mr. Jennings regarding dust losses. At the International smelter at Miami it is noticeable that the bedding bins are entirely housed in, while those at the Calumet and Arizona smelter are exposed to the action of the winds. It would be interesting to hear from Mr. McGregor whether, in this particular, the design at the Miami plant was influenced by the experiencing of excessive dust losses at the Calumet and Arizona plant.

A. G. McGregor, Warren, Ariz.—Of course, we realized that there must be some losses from the wind blowing over the beds. At the International Smelting Co.'s plant at Miami, we have a very valuable material to handle, and we wished to provide against all losses as far as possible, so we housed in the beds.

The Chairman.—The question seems to be largely a commercial one, that is, how great expenditures are justified in order to increase the savings in dust and fume. A plant smelting a low-grade furnace charge can hardly, for the saving obtained, afford to install a process such as that at the International smelter at Miami, where the charge will run from 25 to 35 per cent., without offsetting the savings obtained by an increased charge which would exceed the commercial profit from the operation.

The point which Mr. Jennings makes, that there may be a considerable mechanical loss of fines from the bed systems in the Southwest during the high winds of Spring, is well taken and it might and probably would pay to enclose the ore beds to obviate or reduce this loss.

The only Cottrell plant constructed in Southwestern smelters is that of the International smelter, and I hope Dr. Ricketts can give us some of the results of this installation.

L. D. Ricketts, New York, N. Y.—Mr. Chairman, I thoroughly believe that smelters should not allow their ore to drop through the air in the open and exposed to the wind. The bed should be entirely covered, a sprinkler used if necessary and provision made that the men need not go where the beds are being formed, if the dust is bad. At Inspiration we housed in the beds on account of the richness of the material and used a "V" shaped bin instead of a flat surface and a reclaimer. Mr.

McGregor designed these housed-in bins, but as far as I know, the idea was originally suggested by Mr. Flynn.

In regard to the Cottrell Treaters, I had an article in the *Engineering and Mining Journal** on this subject recently. Apparently the loss of values at Inspiration, where the Cottrell system is used, is nil. Our unaccounted loss probably comes from filling and discharging calcine cars and in turning down the converter and turning it up with the blast on. The dust losses from the reverberatory furnace, even with our method of feeding, are so small that we do not feel justified in putting in treaters. As I have said, the most serious dust loss is from converters and from handling calcines. Our tests show our unaccounted loss at these works to be 0.7 per cent. The values in converter smoke appear to be about 50 lb. of copper per day. The dust loss from the Wedge furnace gases is negligible. The converter-stack losses equal less than 0.2 per cent. of the copper charged to the reverberatory.

* Vol. 102, No. 9, p. 396 (Aug 26, 1916)

An Investigation into the Flowing Temperatures of Copper Mattes and of Copper-Nickel Mattes

BY G. A. GUESS,* M. A., TORONTO, ONT., AND F. E. LATHE,† B. A., M. S., GRAND FORKS, B. C.

(Arizona Meeting, September, 1916)

THIS investigation was started with the idea of determining whether copper-nickel mattes might not have a lower flowing temperature than copper mattes, and thus perhaps aid in accounting for the difficulty so far experienced in attempts at pyritically smelting copper-nickel ores.

We have used the term flowing temperature to avoid confusion with the usual melting temperature. The idea at first was that there might be a greater viscosity range in the one matte than in the other. In the early experiments it was in our minds to determine the relative fluidity of the mattes at a given temperature. Looking to this end we tried pouring equal quantities of matte at the same temperature down an inclined graphite slab, and comparing the distances traveled. Results from such experiments were not satisfactory. We next tried pouring equal quantities at the same temperature through a drilled hole in an Acheson graphite crucible, timing the flow with a stop watch, the whole operation being carried out in the muffle. The flowing temperatures of the mattes tried in this way appeared to be about the same, and there was no marked difference in their fluidity when well fused. Oxidation gave some trouble.

These experiments finally led to the adoption of the following apparatus which we thought at the time was wholly original with us, but which we have since found was described by W. McA. Johnson.[1] The tray used was a slab of graphite 8 in. long, 2 in. wide and 1/4 in. thick. It had six holes, into each of which sat a cup of Acheson graphite. Centrally in the bottom of each cup was a vertically drilled hole 3/16 in. in diameter and 1/2 in. long. The mattes to be tested were put into the cups, in a solid piece. The cups were covered with inverted Battersea "A" cups. The tray was put into the muffle of a gas-fired furnace. Supports on each end held the tray about 2 in. from the floor of the muffle. Under

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[1] *Trans.*, vol. 44, p. 141 (1912).

the apparatus was a layer of fine silica sand spread out smoothly. On such a surface, it was easy to note the falling of the drops of matte. A carefully calibrated thermocouple was used. The junction was inserted under one of the central Battersea crucibles, and rested against one of the graphite cups. The position of the thermocouple was important, as the gradual oxidation of the graphite slab kept it at a temperature considerably above that of the muffle itself. The temperature of the furnace

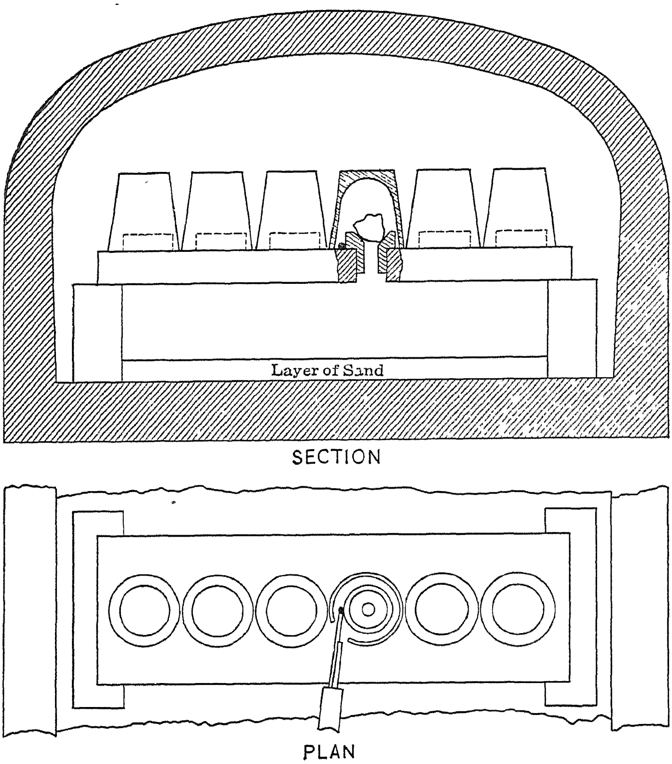


FIG. 1.—APPARATUS FOR DETERMINATION OF FLOWING TEMPERATURES.

was raised gradually, until matte had flowed from each of the cups, and the corresponding temperatures were recorded (see Fig. 1).

Two series of mattes were made by mixing white metal from copper converters and white metal from copper-nickel converters with varying amounts of iron sulphide sticks. These mixtures were fused. The flowing temperatures of the mattes in these series were determined in the manner above described.

Some specimens of furnace mattes which were on hand were likewise tried for their flowing temperatures, and it was found that the flowing temperatures of the furnace mattes were in every case higher than the

flowing temperatures of the laboratory-made mattes of the same grade. This was true both for copper mattes and for copper-nickel mattes. About this time it was noticed that some of the low-grade laboratory

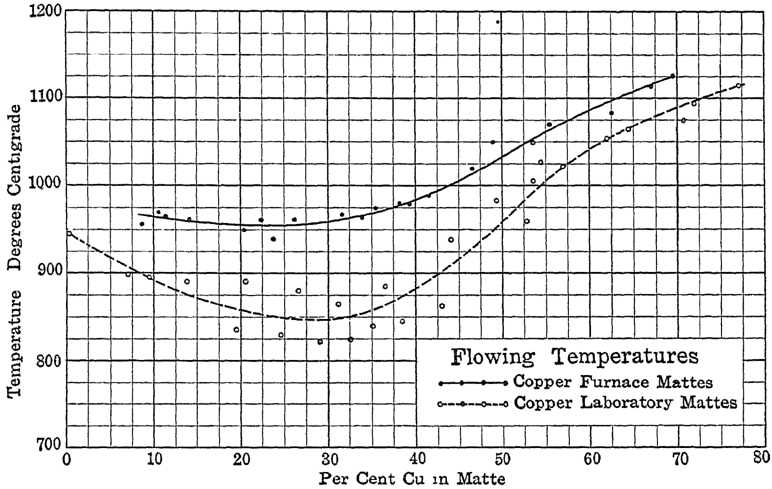


FIG. 2.—COMPARATIVE FLOWING TEMPERATURES OF COPPER FURNACE MATTES AND COPPER LABORATORY MATTES.

mattes were disintegrating. They had never looked just like furnace mattes, having from the first a yellow tint, but when they started to disintegrate our suspicions were further aroused, and these suspicions

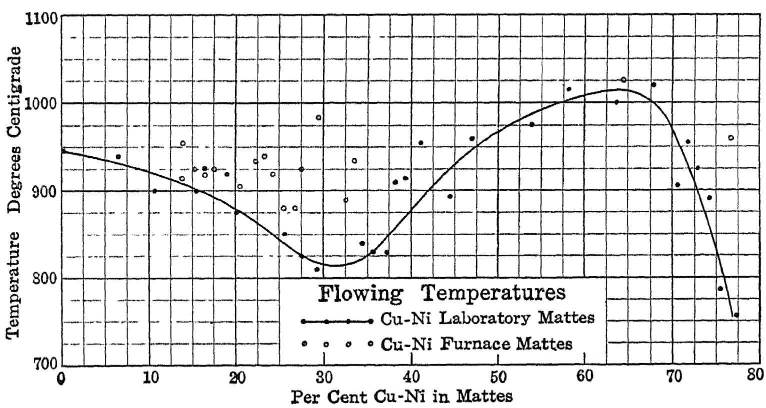


FIG. 3.—COMPARATIVE FLOWING TEMPERATURES OF COPPER-NICKEL LABORATORY AND FURNACE MATTES.

were confirmed when the sulphur content was found to be abnormal. For example, the sulphur in a laboratory-made matte containing 16.3 per cent. copper was found to be 31.2 per cent., nearly the theoretical;

whereas the sulphur in a furnace matte of the same grade was 26.0 per cent.

This finding discredited the whole series of mattes and led us to believe that the disagreement in the melting temperatures of mattes as found by various investigators was due to the difference in conditions under which the mattes were prepared, and that it might be well to distinguish between furnace mattes and laboratory-made mattes.

We next procured, through the courtesy of some smelters, several furnace mattes of different grade. We found them all to have a higher flowing temperature than laboratory mattes of the same grade. This difference was more pronounced in the mattes of low and medium grade.

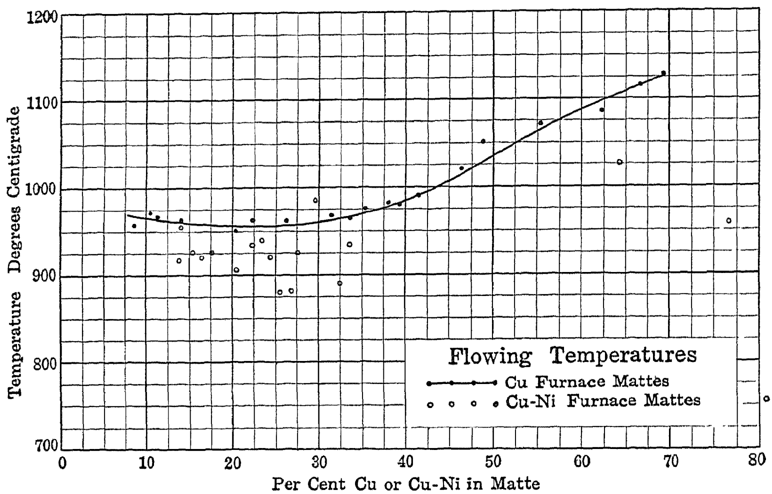


FIG. 4.—COMPARATIVE FLOWING TEMPERATURES OF COPPER AND COPPER-NICKEL FURNACE MATTES.

The effect of remelting furnace mattes under borax glass and under charcoal was tried. When melted under borax, furnace mattes became apparently laboratory mattes having the same sulphur content and flowing temperature. The fact that borax glass dissolved a considerable amount of iron from the mattes, suggests that furnace mattes contain iron oxides, as has been remarked by several writers.

Curves have been plotted showing the flowing temperatures of the two series of laboratory-made mattes, of a series of furnace copper mattes, and of a partial series of furnace copper-nickel mattes (Figs. 2 to 5).

The results of these experiments have not shown as great a difference in flowing temperature between copper and copper-nickel mattes as was expected. Copper-nickel mattes appear to have a flowing temperature from 30° to 50° below copper mattes of the grade usual in blast-furnace work. This slight difference in temperature does not appear to us to be

sufficient to explain failures in pyritic smelting due to a too early liquation of copper-nickel matte in the furnace.

The determination of flowing temperature as shown by the accompanying curves indicates the existence of a eutectic containing about 27 per cent. copper or copper nickel. This is seen much more clearly in the curves of laboratory-made mattes.

The following tables show the results of melting some furnace mattes under borax glass and under charcoal, and the copper and nickel content of the copper-nickel mattes used.

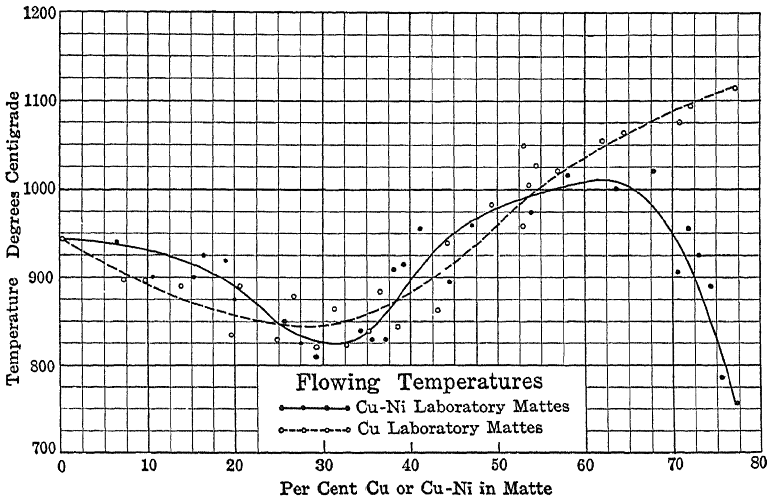


FIG. 5.—COMPARATIVE FLOWING TEMPERATURES OF COPPER-NICKEL AND COPPER LABORATORY MATTES.

Copper Mattes

No.		Per Cent. Cu	Per Cent. S	Flowing Temperature °C.
21	Original furnace matte..........	10.8	24.2	970
37	No. 21 melted under borax glass..	13 6	29.9	890
21*c*	No. 21 melted under charcoal....	11.8	25.5	850

Copper-Nickel Mattes

No.		Per Cent. CuNi	Per Cent. S	Flowing Temperature °C
40	Original furnace matte..........	13.7	26.4	912
22	No. 40 melted under borax......	16.34	31.3	930
40*c*	No. 40 melted under charcoal....	15.25	27.2	900

Copper-Nickel Furnace Mattes

Per Cent Cu	Per Cent. Ni	Per Cent CuNi	Flowing Temperature °C.
4 00	9.70	13.70	912
4 05	9.65	13.70	955
4.45	10.65	15.10	925
4.65	11.65	16 30	920
5 15	12.60	17.75	925
5 95	14.30	20 25	905
6 40	15.75	22 15	935
6 95	16.20	23 15	940
7.20	17.00	24.20	920
7 40	17 75	25 15	880
8 25	18 30	26.55	880
6 10	21 50	27 60	922
8.65	20 85	29 50	985
9 20	23 30	32 50	890
9.35	24 10	33 45	935
19.05	45.20	64.25	1,027
24.90	51 85	76 75	960
22 40	58 40	80.80	757

Copper-Nickel Laboratory Mattes

Per Cent Cu	Per Cent. Ni	Per Cent. CuNi	Flowing Temperature °C.
1 70	4 70	6.40	940
2 95	7.51	10 46	900
4 19	11 06	15.25	900
4.82	11.52	16 34	924
5.98	13.00	18.98	920
6 45	13.64	20 09	874
7 98	17.44	25.42	850
8 25	19.07	27.32	826
9.18	20.06	29 24	810
10 56	23.64	34.20	840
10 52	24.90	35 42	830
11 89	25.24	37 13	831
11 22	26 60	37.82	913
12 12	27 06	39 18	915
12.40	28 76	41.16	955
13 72	30.58	44 30	894
15.36	31 44	46.80	960
16.49	37.35	53.84	975
16.60	41.46	58.06	1,015
19.36	44.20	63.56	1,000
20.00	47.49	67.90	1,020
22.80	47.60	70.40	905
21 91	49.93	71.84	957
28 50	44.40	72.90	925
28.80	45 60	74.40	890
23 87	51.43	75.30	785
20 40	56.72	77.12	755

Features of the New Copper Smelting Plants in Arizona

BY A. G. McGREGOR,* B. S., M. E., WARREN, ARIZ.

(Arizona Meeting, September, 1916)

During the past 5 years, five new copper-smelting plants have been built and put into operation in the State of Arizona. The monthly copper output from these plants averages from 5,000,000 to 18,000,000 lb. Previously, there never was as much activity in copper-smelting plant construction in the same length of time in all the rest of the world.

Naturally, in this amount of work, some new problems were met and new features in plant design and equipment developed. Some of these features are described in this paper.

Concentrate Hauling, Unloading and Sampling System

(a) *Concentrate Car*

The International Smelting Co. has provided at its plant at Miami specially designed cars for hauling the concentrates from the mills of the Inspiration Copper Co. and the Miami Copper Co. The car has a bottom that is readily made tight to hold flotation concentrates and is quickly and cheaply unloaded. Figs. 1, 2 and 3 show views and details of the car.

The main feature of this car is the slot in the bottom throughout its length, which is closed by means of short planks that overlap like ship-lap as shown by Fig. 1. At one end of the car an iron gate operated by a screw and hand wheel is provided, which clamps or presses the planks together, making the bottom tight. A removable tapered plug rests vertically over this gate when the loading of the concentrates at the mill commences.

A concentrate unloading pocket (Fig. 4), 180 ft. long, is provided at the smelting plant, which has a conveyor belt running throughout its length underneath the track. To unload a car, the vertical plug at the end of the car is lifted by means of a chain hoist which is supported by a trolley overhead. The gate at the end of the car is opened approximately 18 in. The removal of the plug leaves a hole down through the concentrates. The concentrates are poked through the hole from the top of the car until it is convenient for a man to stand on the bottom and draw one of the slats in the bottom toward the gate end of the car. The concentrates are then raked down with hoes on their natural angle of repose,

* Mechanical and Metallurgical Engineer.

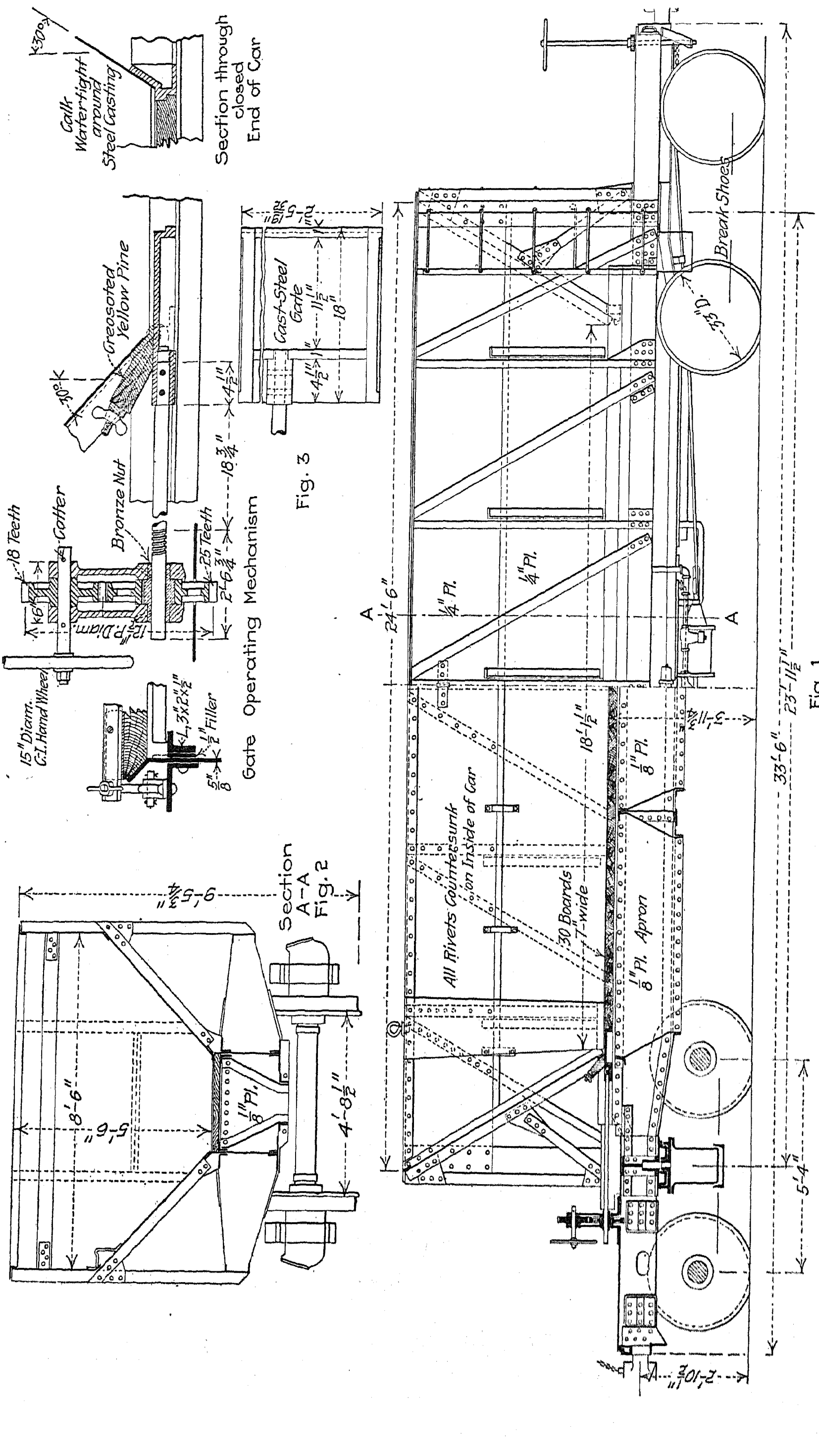


FIG. 1.—6-TON CONCENTRATE CAR. FIG. 2.—SECTION A-A OF FIG. 1. FIG. 3.—GATE-OPERATING MECHANISM OF CONCENTRATE CAR.

which is nearly vertical, until the next slat is partly uncovered, when it is pulled toward the gate end of the car. This operation progresses until the opposite end of the car is reached. The slats and the grooves supporting them are then carefully swept off and cleaned with a broom; the gate is screwed back, pressing the slats together, and the plug is lowered into place. The car is then ready to be taken back to the concentrator for another load.

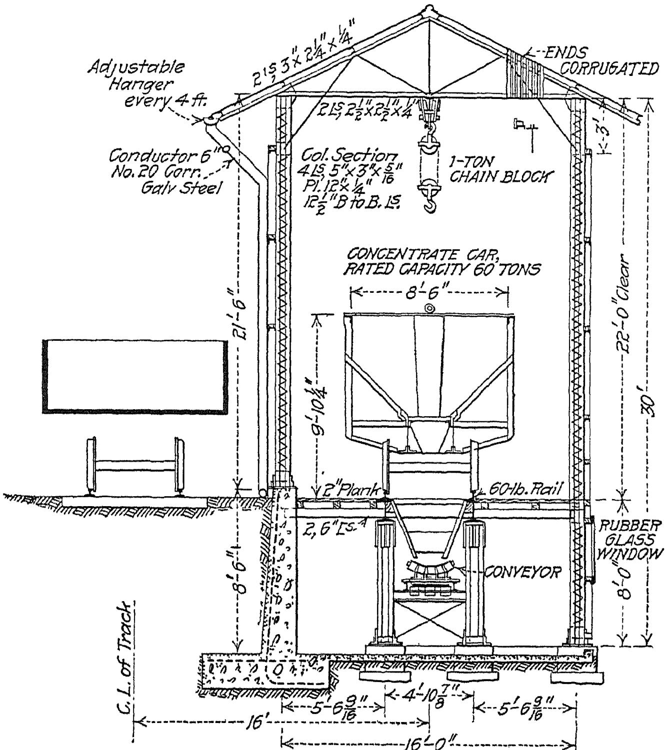


FIG. 4.—SECTION THROUGH CONCENTRATE UNLOADING POCKET.

(*b*) *Concentrate Sampler*

Fig. 5 shows diagrammatically the scheme used for sampling the concentrates. It will be noted that the conveyor from the unloading pocket discharges onto a shuttle conveyor (a tripper on the former conveyor could have been substituted for the shuttle conveyor but it would not have handled the sticky concentrates as well). Buckets attached at their ends to chain belts are arranged to cut through the stream of concentrates as they are discharged onto the shuttle conveyor. The chains are driven and supported by sprocket wheels. The arrangement is such that at the highest point in their travel the buckets are in an inverted position. At

this point, they pass under a revolving shaft which has a number of knockers, made of 6-in. belting, attached to it. These knockers slap the bottom of the buckets, jarring loose any concentrates tending to stick to them. This arrangement works satisfactorily on ordinary table and vanner concentrates, but is not satisfactory with flotation concentrates on account of the varying load on the conveyor belt, the flotation concentrates being unloaded from the car in large chunks.

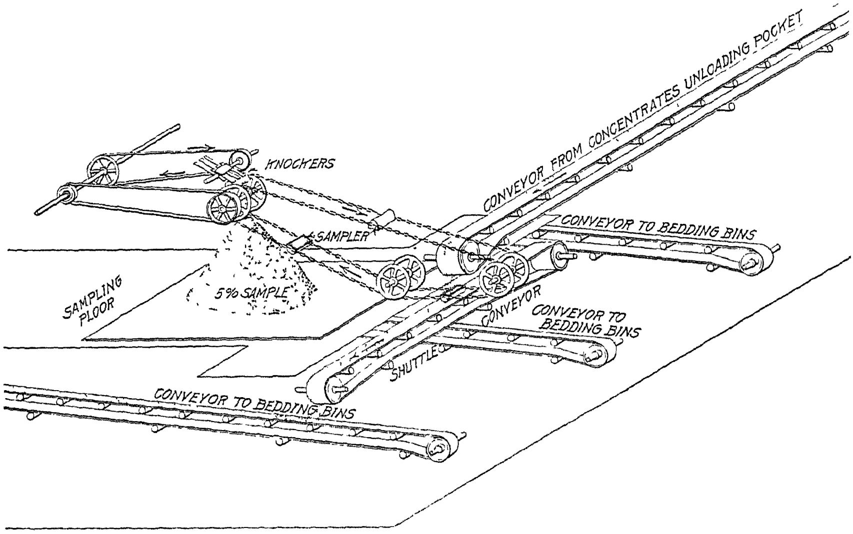


FIG. 5.—ARRANGEMENT FOR SAMPLING CONCENTRATES.

(c) *Bedding Bins*

Fig. 6 shows a cross-section of the bedding bins used in connection with the concentrate car and sampler just described. The bins are 150 ft. long and have a total capacity of 9,000 tons of charge. The concentrates are delivered by belts from the car-unloading pocket to any one of the three conveyors passing over the bins. Each belt has a motor-operated tripper which travels back and forth throughout the length of the bin. Each tripper is arranged with two cross-belts, which discharge material on both sides of the tripper, instead of the ordinary chutes that would clog with the sticky concentrates. These cross-belts also permit of a wider top on the ore bed than could be obtained with ordinary chutes. The cross-belts of the tripper as well as the long belts running over the bins and all the other belts handling sticky concentrates are speeded to 400 ft. per minute, which aids in discharging the sticky material from the belt when it passes over the pulley at its head end or the pulley of the tripper. An oscillating deflector is so

arranged that the material passing over the tripper is discharged, first on one of the short cross-belts and then on the other. This is done to provide an even distribution of the material over the top of the beds, making the height of the bed on both sides of the tripper the same. The tripper is reversed at each end of its travel by the reversal of its propelling motor through a magnet-switch control. The bins have "V" bottoms with a slot 2 ft. 4 in. wide running throughout their length. This slot is covered with short planks which are readily removed, and as the reclaiming progresses, they are merely moved back in the slot.

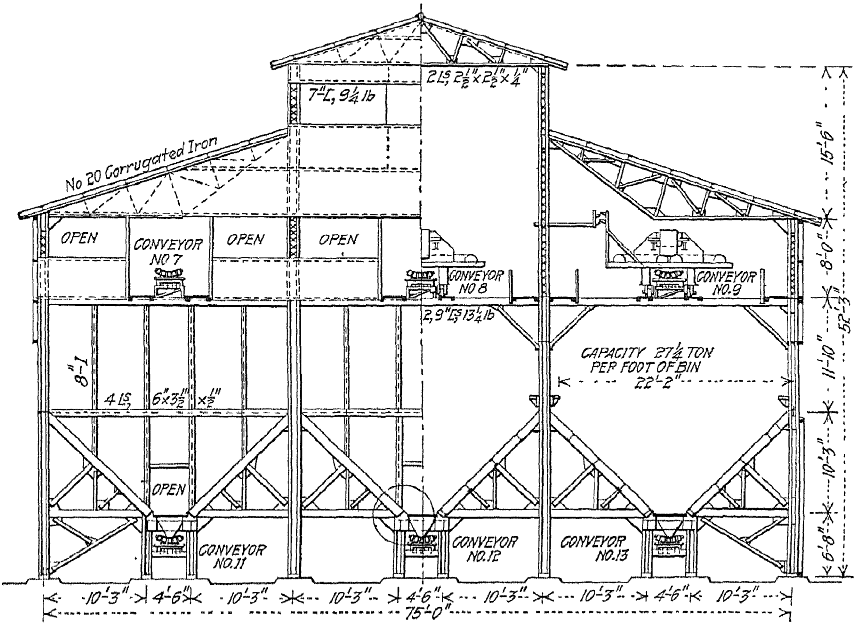


FIG. 6.—CROSS-SECTION OF BEDDING BINS.

The concentrates are raked down with hoes and allowed to discharge, through the opening in the bottom thus provided, onto the conveyor belts underneath. The concentrates are bedded in these bins with a proper amount of limestone and pyrite and the available secondaries, each bin comprising a bed. Storage bins for storing finely crushed lime rock, secondaries, pyrite, etc., are provided and arranged so that the fine material from these bins is delivered by a conveyor and discharged just ahead of the concentrates onto the belts leading up over the bedding bins. The material next to the bedding belts, therefore, consists largely of finely crushed lime rock, which is a further aid in the discharge of the sticky concentrates from these belts.

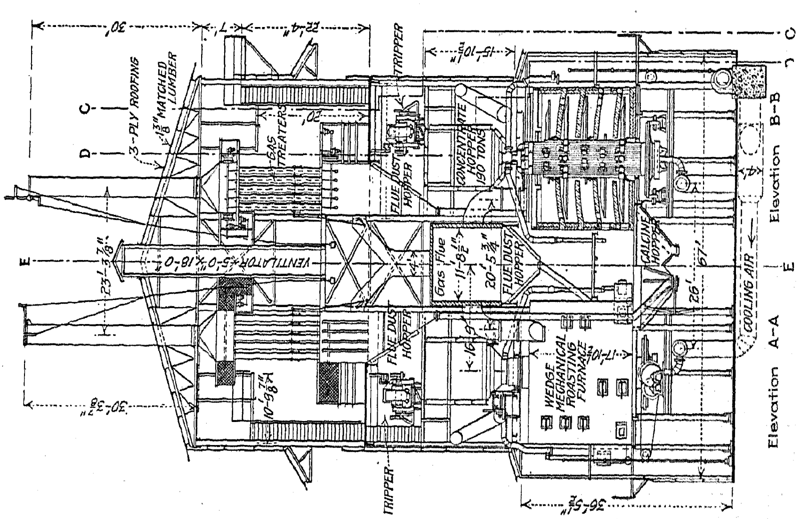


FIG. 7.—END ELEVATION OF ROASTER OR DRYER PLANT, INTERNATIONAL SMELTING CO.

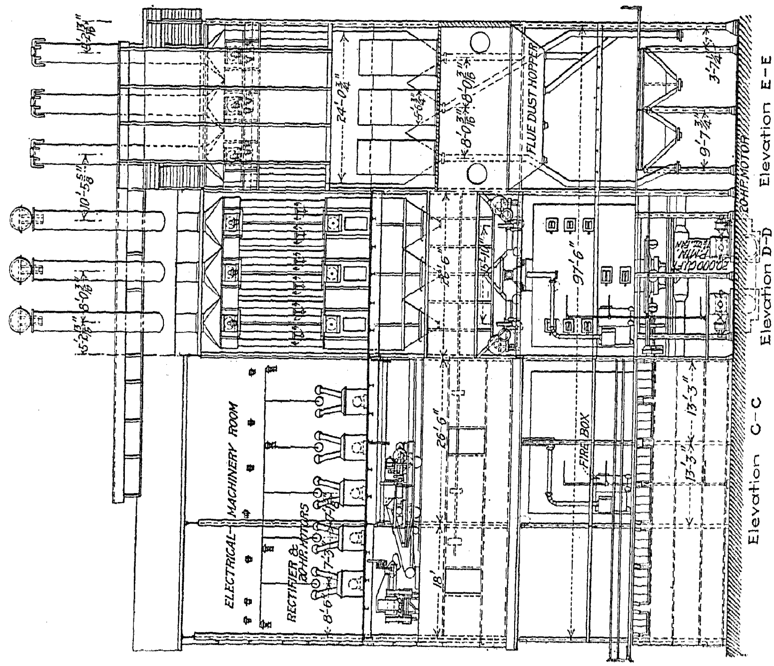


FIG. 8.—SIDE ELEVATION OF ROASTER OR DRYER PLANT, INTERNATIONAL SMELTING CO.

Roaster or Dryer Plant

Figs. 7 and 8 show end and side elevations respectively of the International Smelting Co.'s roaster or dryer plant. The main features of this plant consist in the method used for conserving the heat and in the placing of the Cottrell fume treaters directly above the furnaces. The roaster furnaces in this plant are used for drying the concentrates and are not used for roasting, as the charge contains no excess sulphur; in fact it has been necessary to have some pyrite shipped in from Bisbee in order to raise the sulphur content of the charge sufficiently to prevent the production of mattes too high in copper for good clean slags.

(a) *Conservation of Heat in Furnace*

The furnaces are air-cooled. The cooling air, after passing through the furnace arms, is conducted up the central shaft to the top, thence down to the oil fireboxes connecting with the two lower hearths of the furnace. This gives preheated air for the combustion of the oil and conserves the heat absorbed by the cooling air. To further conserve the heat of the furnace, the linings are 12 in. thick. The outer 4 in. of lining, between hearths, is made of "Nonpareil" insulating brick. The manufacturers of these brick claim that 4 in. of their brick is equivalent in heat insulating value to 40 in. of ordinary brick. The tops of the calcine hoppers of these furnaces are insulated from the lower floor of the furnace building, just above them, by an air space. The calcine hoppers have non-conducting linings consisting of 2½ in. of Nonpareil insulating brick and a 1-in. layer of reinforced concrete. The reinforced concrete is to take the wear and erosion caused by the heated material inside the hopper, as the Nonpareil brick will not stand much wear or rough usage.

(b) *Cottrell Dust-collecting System for Furnaces*

The Cottrell equipment for treating the gases from this plant consists of 12 units of 20 tubes each. The tubes are of lap-welded steel 13-in. in outside diameter, and 15 ft. in length. They are flanged outward, or "Vanstoned" at each end, which reduces the brush discharge caused by sharp corners and makes a good means of connection to the upper and lower diaphragms. Suitable lugs are welded to the pipes against which the hammers strike when vibrating the tubes to shake off the collected dust. The electrical equipment for this installation is so arranged that the voltage used may be varied from 50,000 to 100,000 volts. The treater tubes were planned for a velocity of 5 ft. per second for the gases passing up through them. The recovery of dust by this treater is practically perfect.

The main feature of this installation is in its arrangement which permits the gases to pass directly up from the furnaces through the header flue, the tubes and the stacks, the gases rising continuously from the furnace hearths to the outlet of the stacks.

Blast Furnaces

Fig. 9 shows a general end elevation of the blast furnaces at the Calumet & Arizona Mining Co.'s plant at Douglas. These furnaces are

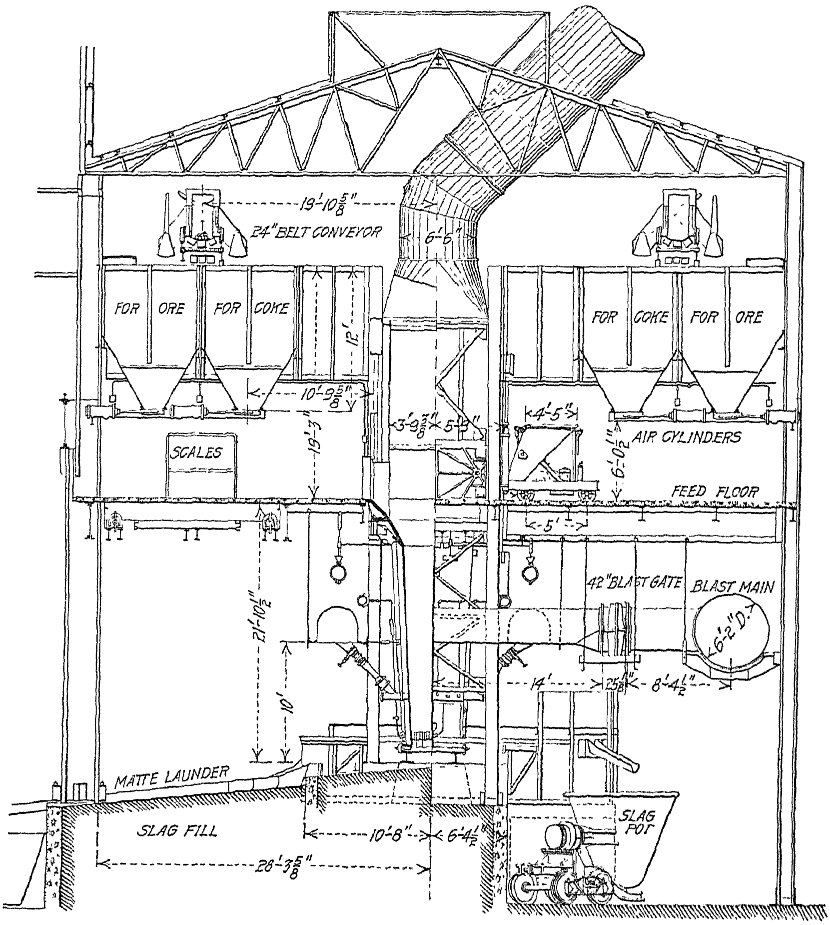


Fig. 9.—End Elevation of Blast-furnace Plant, Calumet & Arizona Mining Co., Douglas.

40 ft. long by 4 ft. wide at the tuyères. The ores and materials smelted in these furnaces are bedded so that the charge coming to the charge bins over the furnaces is thoroughly mixed. A coke bin and an ore bin are pro-

vided on each side of each furnace. Each furnace has four charge cars, two on each side. The cars are 20 ft. long. When receiving a charge of ore or coke, the cars are resting on track scales. Each car has four compartments and the bins overhead have a gate corresponding to each compartment, so that the amount of charge for any part of the furnace can be regulated to suit the conditions. The car is propelled from the bins to the charging position at the furnace, a distance of 7 ft. or 15 ft., (according to whether a charge of coke or a charge of ore is being moved) by means of an electric motor geared to two wheels of the car. Fig. 10 shows a view of the charge car under the charge bins.

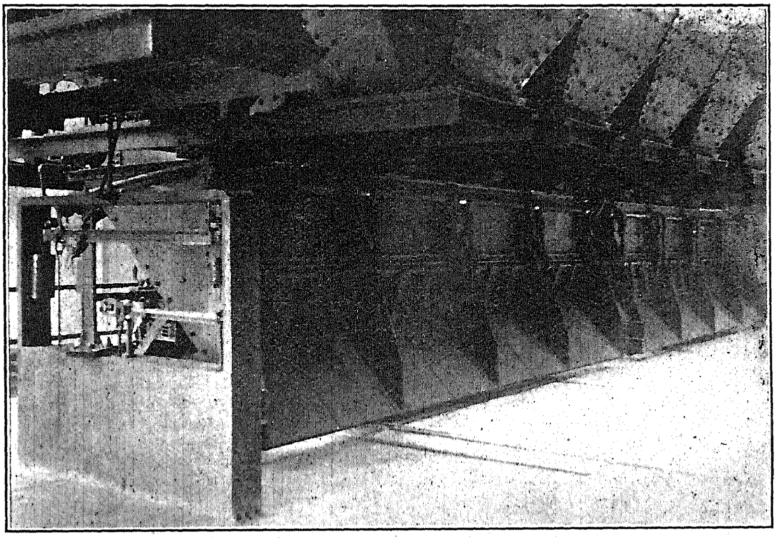


FIG. 10.—VIEW SHOWING CHARGE CAR UNDER BINS.

The furnace side jackets are in one length 16 ft. 6 in. long and are 40 in. wide. It will be noted that the bustle pipe is combined with the truss or girder which takes the outward thrust of the jackets at their midpoint. This simplifies the construction and gives more room around the bustle pipe for the water piping, etc. Fig. 11 is a view showing the tuyères and bustle pipe.

The bridge plates for the furnace bottoms are of structural steel; ½-in. plates are riveted to 6-in. I-beams. The I-beams are spaced 12½ in. apart. The cooling of the bottom is effected entirely by the surrounding air. The I-beams riveted to the plates undoubtedly increase the effectiveness of the cooling. The I-beams supporting the bottom are also used for holding the outward thrust of the bottom ends of the jackets. This thrust is carried into the beams by means of steel rods threaded at one end and flattened at the other. The flattened ends are riveted to the

webs of the beams and a nut on the threaded end holds a lug against the bottom end of the jacket.

The furnace tops and charge doors are of structural steel in the form of air jackets. An air space is provided between the inner and outer sheets of the furnace tops. The lower end of this air space connects with the outside air while the upper end connects with the inside of the furnace top. The cool air is drawn in at the bottom and discharged at the top into the furnace gases going to the chimney, thus insuring a positive air circulation which keeps the sheets of the air top properly cooled. The furnace charge doors are jacketed in a similar manner.

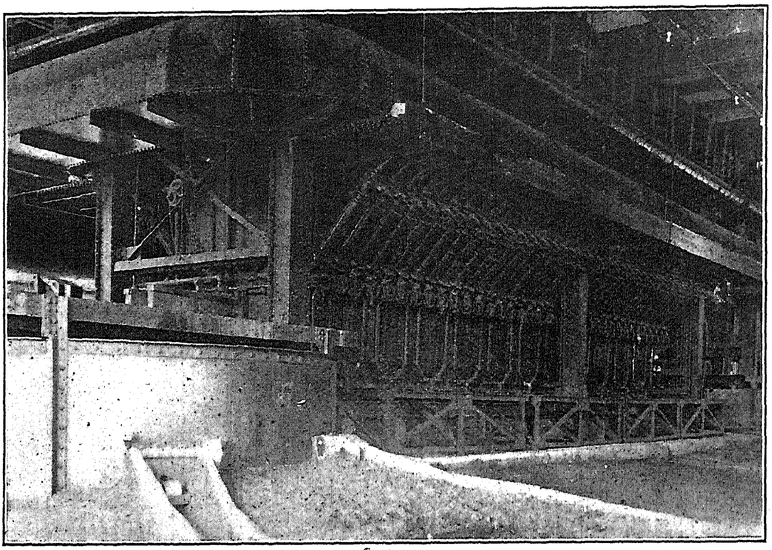


Fig. 11.—View Showing Tuyères and Bustle Pipe, Blast Furnace of Calumet & Arizona Mining Co., Douglas.

These furnaces have been in use for 3 years, smelting as high as 1,400 tons of charge each per day, and no repairs or replacements have been made in furnace water jackets and no warping has taken place in the air-cooled tops.

Reverberatory Furnaces

Fig. 12 shows a side elevation of a typical reverberatory furnace at the International Smelting Co.'s plant which has several features worthy of mention.

(a) *Furnace Bottoms*

It was especially desirable at this plant, on account of difficulties at another plant, to avoid any trouble with furnace bottoms in starting up.

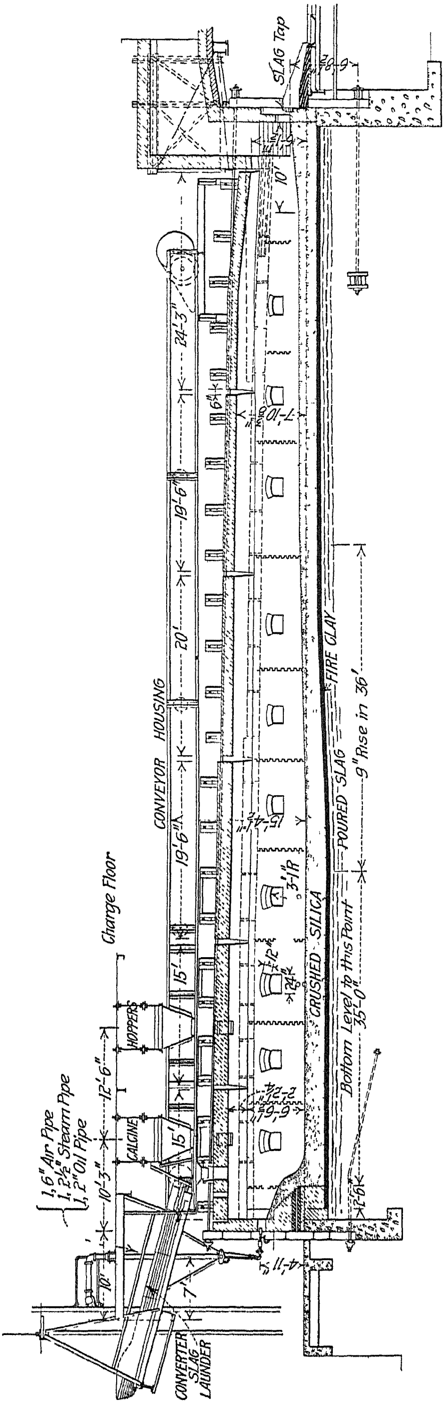


FIG. 12.—LONGITUDINAL SECTION OF REVERBERATORY FURNACE, INTERNATIONAL SMELTING CO.

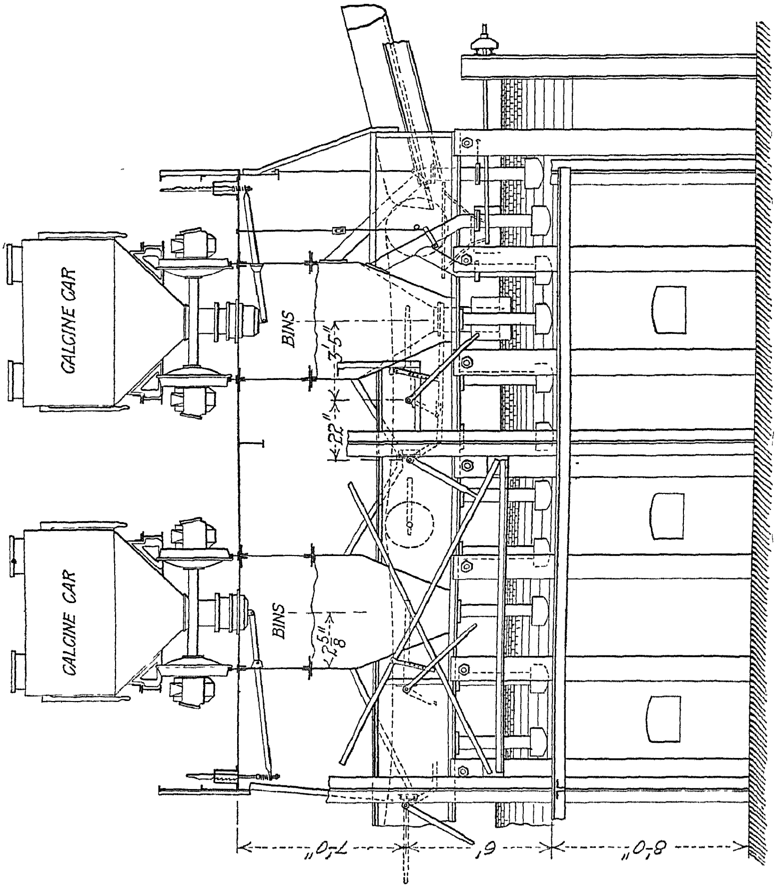


FIG. 14.—SIDE ELEVATION OF REVERBERATORY FURNACE SHOWING CHARGING ARRANGEMENT.

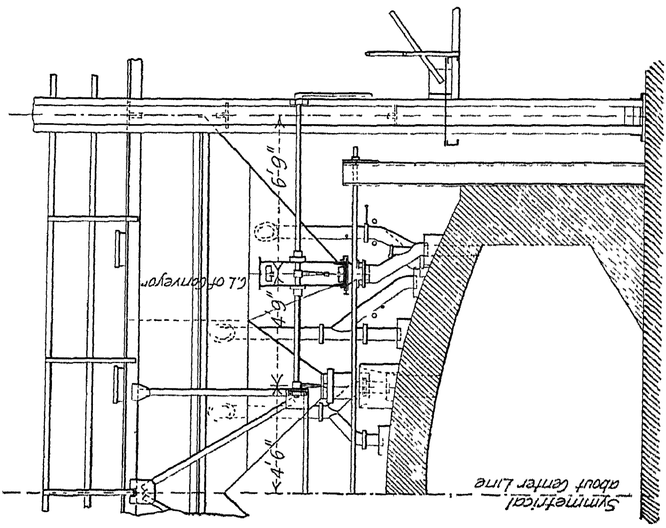


FIG. 13.—SECTION REVERBERATORY FURNACE SHOWING CHARGING ARRANGEMENT.

To accomplish this, broken slag, brought in railroad cars from the Old Dominion smelter, was melted in a small blast furnace, obtained by setting up a number of discarded furnace jackets and a motor-driven blower, on the site for the reverberatory furnaces. The molten slag thus obtained was conducted in launders to the foundations of the reverberatory furnaces. Heavy concrete beams and struts were provided between the furnaces for taking the thrust from the lower ends of the buck stays. Later, the spaces between the concrete struts and the beams were filled in with molten slag obtained from the regular operation of the reverberatory furnaces. The slag bottom was covered first with a 4-in. layer of fire clay and then a 27-in. layer of silica (94 per cent. SiO_2) crushed to minus ¼ in. These bottoms gave no trouble whatever in starting up.

(b) *Furnace Fettling and Charging System*

The method of charging, instead of fettling only, along the side walls of reverberatory furnaces, now so successfully used by the Canadian Copper Co. and the Anaconda Copper Mining Co., was developed after the construction of this plant was well along. The plant was laid out for charge tracks running at right angles to the furnaces near the firing end. In order to distribute the charge along the sides of the furnaces, charge hoppers were located under the charge tracks directly over the side walls of the furnaces. Drag-chain conveyors were installed, one over each side of each furnace, which received the charge from the charge hoppers (see Figs. 13 and 14). Under these conveyors, approximately every 30 in., suitable down-spouts with gates are provided so that the charge may be distributed along the side walls of the furnaces throughout their length. The bridge wall and side walls at the firing end of the furnaces are charged by drawing directly from the hoppers through suitable spouts.

The floor over the charge tracks around the skimming end of the furnace is paved with firebrick.

(c) *Calcine Cars*

Figs. 15 and 16 show end and side elevations respectively of the calcine car used at the International Smelting Co.'s plant. The feature of this car is the arrangement used for reducing dust losses in receiving and discharging a charge. The car is equipped with four sliding sleeves in the top which are spaced to register with the discharge openings of the calcine hoppers at the roaster plant. When the car is spotted under the roaster hoppers in its proper position, the sleeves are forced up against the calcine hoppers by means of levers on the side of the car. The operating lever is of flat spring steel so that the sleeve can be firmly pressed into con-

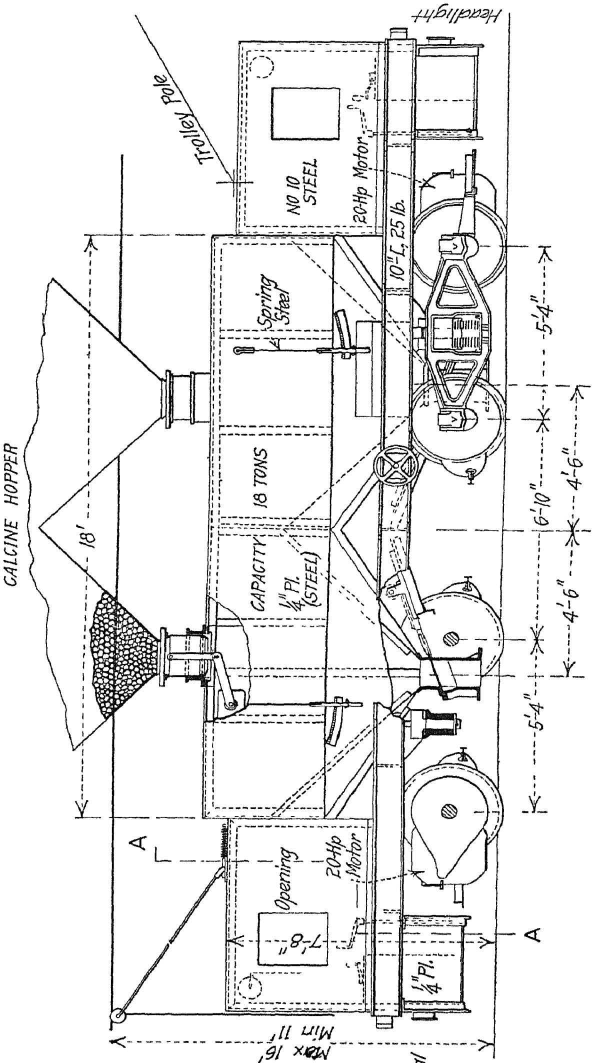


FIG. 16.—SIDE ELEVATION OF CALCINE CAR.

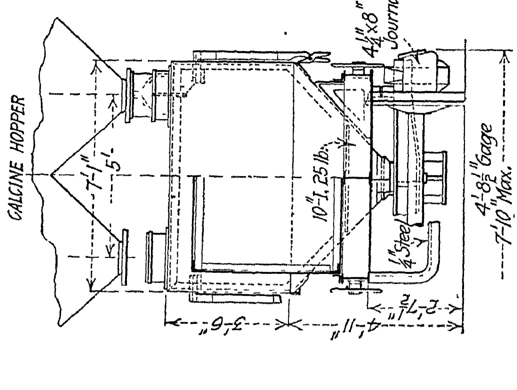


FIG. 15.—END ELEVATION OF CALCINE CAR

tact with the flange at the bottom of the calcine hopper, and held there by a notch in a quadrant on the side of the car. The spring handle insures a good pressure between the sleeve in the top of the car and the flange of the calcine hopper regardless of the small variations in the distance between the top of the car and the calcine hoppers or any lost motion in the connection of the lever with the sleeve. An air vent is provided in the car made of fine woven wire mesh.

Fig. 14 shows the sleeves in the top of the reverberatory furnace charge hoppers for receiving the charge from the calcine car. These sleeves are similar to the sleeves in the top of the calcine car. By pressing down on the lever, which is operated from the charge floor, the sleeve is forced up against the flange of the discharge spout of the car. Notches are provided in the lever which engage a catch and thus hold the sleeves in their upper position while the car is being discharged. The pressure of the handle is transmitted to the sleeve, as will be seen from the figure, through a helical spring, so that a good contact between the sleeve and the spout of the car is assured regardless of the lost motion and wear in the operating levers and the variations in the distance between the sleeves in the charge floor and the flange on the discharge spout of the car.

(*d*) *Baffling in Waste-Heat Boilers*

By reference to Fig. 17, it will be noted that the header flue between the reverberatory furnaces and the waste-heat boilers, and the connections between the header flue and boilers are well up above the furnace floor level, giving ample head room for the convenience and comfort of the workmen; the bottom of the header flue is 10 ft. 6 in. above the furnace floor level. Also, it will be noted that the boiler-room floor level corresponds to the furnace floor level. This is made possible by reversing the baffling in the Stirling boilers so that the gases enter the front of the boiler near the top and leave the boiler at the rear near the bottom instead of as in the standard setting.

This arrangement of baffling gives equally as good results from the standpoint of water circulation in the boiler, priming, etc., as the standard baffling.

Arrangement of Converter Plants

Southwestern practice is to have the converter plant adjacent to the reverberatory furnaces, and arranged so that the reverberatory matte is received through launders into ladles directly under the converter plant crane. Also arranged so that the converter slag may be poured from ladles by the converter crane back into the reverberatory furnaces through launders extending from the converter aisle to openings provided in the roofs of the reverberatory furnaces. The converter slag is usually

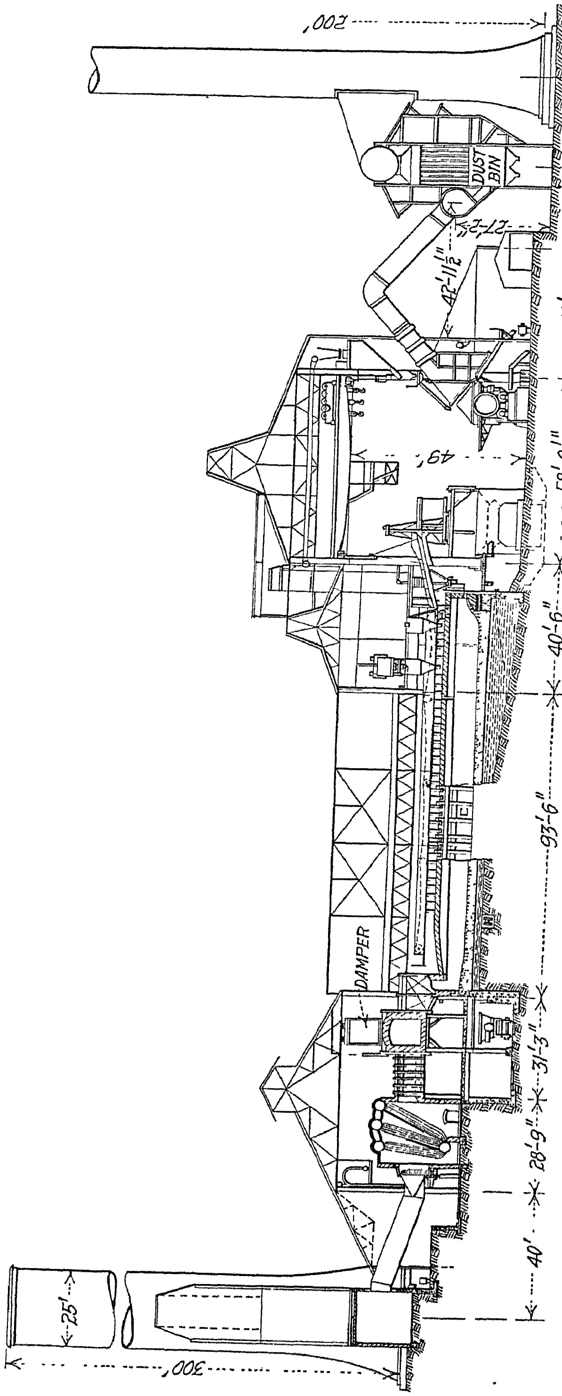


FIG. 17.—TYPICAL CROSS-SECTION OF THE INTERNATIONAL SMELTING CO.'S PLANT.

discharged several feet in front of the bridge-wall of the furnace and midway between the side walls. Fig. 17 is a typical cross-section of the International Smelting Co.'s plant and is typical of the new reverberatory smelting plants in Arizona. Fig. 18 is a plan of the same plant.

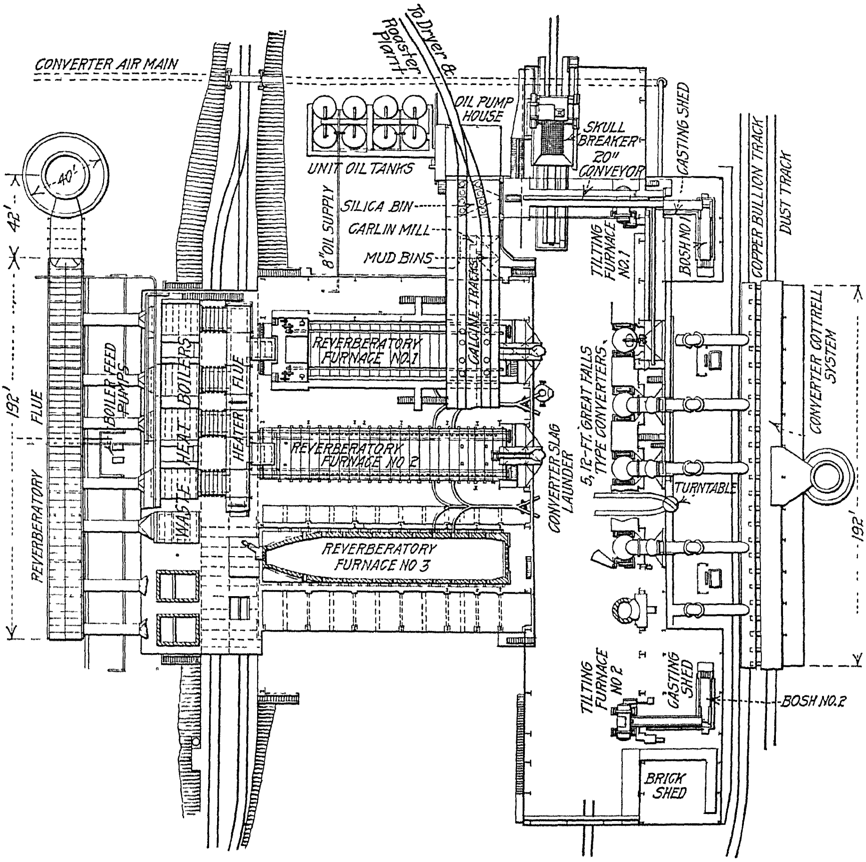


FIG. 18.—PLAN OF INTERNATIONAL SMELTING CO.'S PLANT.

(a) *Siliceous Ore Charging Device for Converters*

Several of the plants are arranged so that the siliceous ore for the converters is drawn directly from bins overhead. Figs. 19 and 20 show a typical arrangement of bin, measuring hopper and chute for accomplishing this operation. This particular arrangement is installed at the International Smelting Co.'s plant. A weighing or measuring hopper is interposed between the overhead bin and the chute leading to the converter mouth. The hopper is mounted on springs and is connected by levers and links to an indicator readily seen from the operating floor. Any desired amount of charge can be weighed out into the measuring hopper

and discharged into the converter and this can be accomplished entirely through the operation of levers on the main floor of the converter plant.

(*b*) *Converter Plant Cranes*

Most of the newer plants are equipped with 12-ft. Great Falls type converters. These converters weigh, when lined, approximately 70 tons. Instead of designing the converter cranes for a capacity to lift

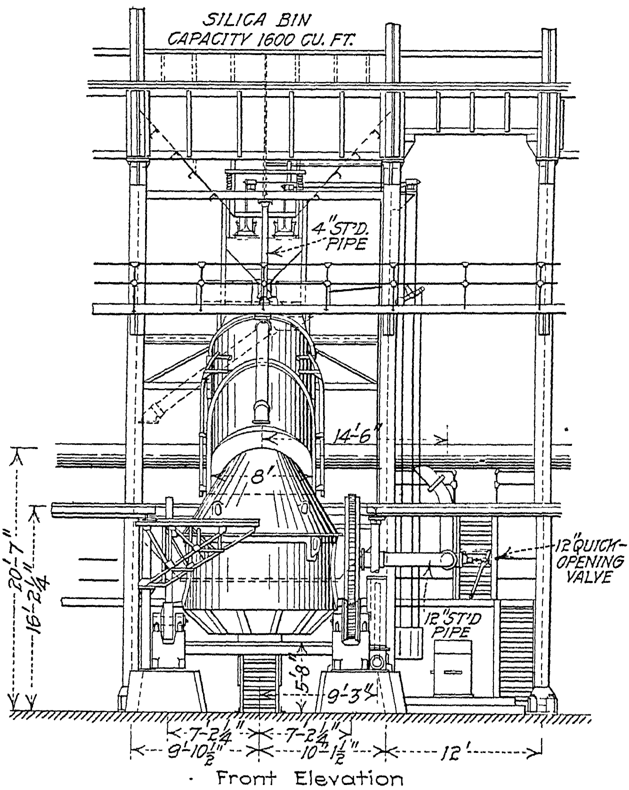


FIG. 19.—CONVERTER PLANT, INTERNATIONAL SMELTING CO.

a converter shell with its lining, as has usually been the practice heretofore, the cranes are designed for a lifting capacity of 40 tons, which is nearer right for the routine work of handling matte, copper and slag in large ladles. Converters with magnesite lining do not have to be moved into and out of their stalls often, but when it is desirable to do this two cranes are used with a lifting rig as shown in Figs. 21 and 22. The converter cranes in all of the Arizona smelting plants are operated by 250-volt direct current. The later cranes are equipped with magnet-switch control.

(c) Skull Breaker for Converter Plant

Three of the Arizona converter plants are equipped with skull breakers and a fourth plant is now being thus equipped. Figs. 23, 24 and 25 show the general arrangement of the skull breaker at the International Smelting Co.'s plant. A hopper about 11 ft. wide by 24 ft. long is provided, the

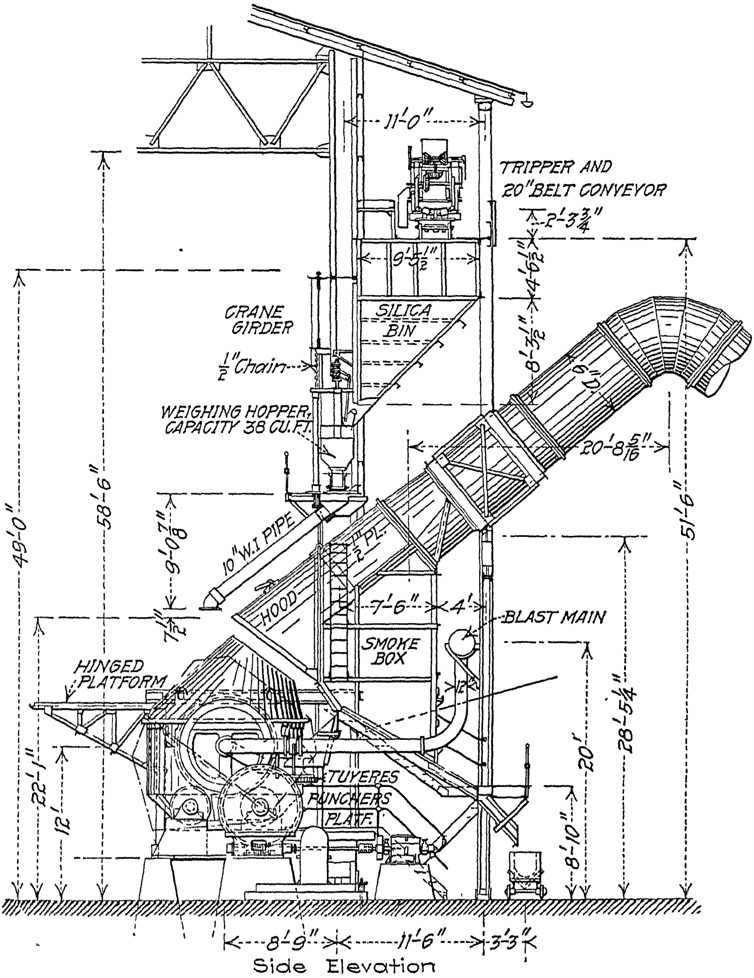


Fig. 20.—Converter Plant, International Smelting Co.

bottom of which consists of cast-steel bars with projecting lugs, making a grating with openings throughout about 9 in. square. A small traveling bridge is provided over this grating, and mounted on this bridge is a trolley which travels transversely to the path of the bridge. In the trolley is mounted a motor-driven hoist which raises a hammer in the form of a long

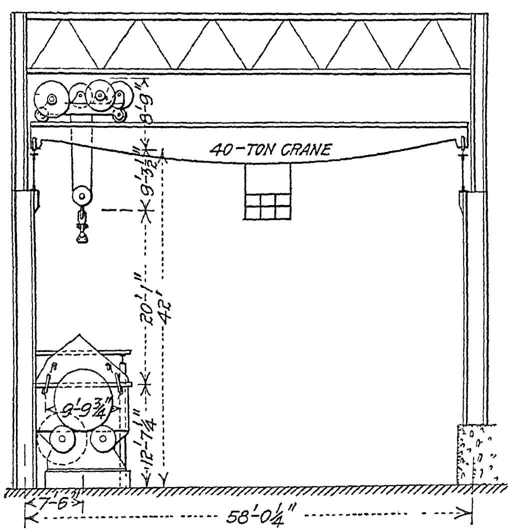


Fig. 21.—Method of Lifting Converters, Calumet & Arizona Mining Co

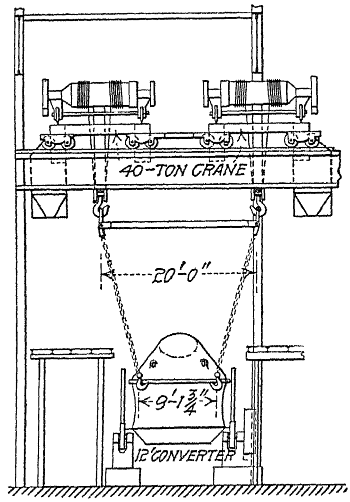
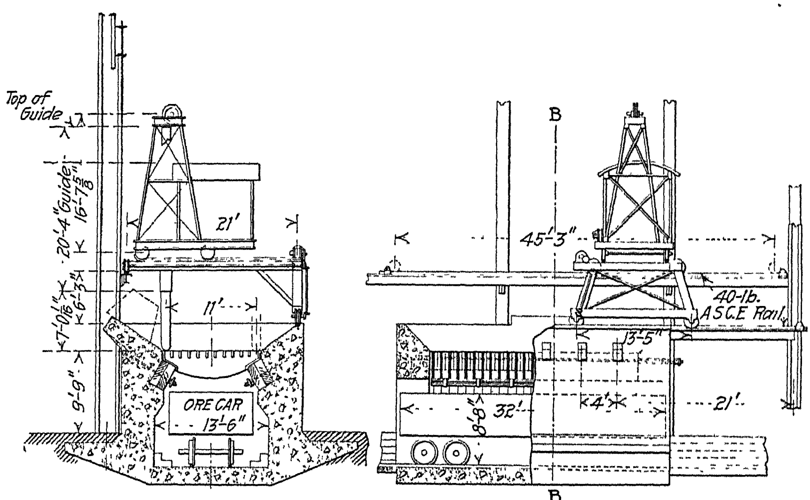


Fig. 22.—Method of Lifting Converters, Calumet & Arizona Mining Co.



Section B-B, Fig. 23 Section A-A, Fig. 24

Figs. 23 and 24.—Skull Breaker, International Smelting Co.

weight that works in a guide. The hammer and guide are mounted on the trolley with the hoist. The hoist operator raises the hammer and allows it to drop on the skull in a manner similar to the operation of the ordinary type of pile-driver hammer. When the skulls are broken into fragments sufficiently small to pass through the grating, they drop into a steel railroad car underneath. The bridge and trolley travel permit the hammer to be spotted over any skull on any part of the grating.

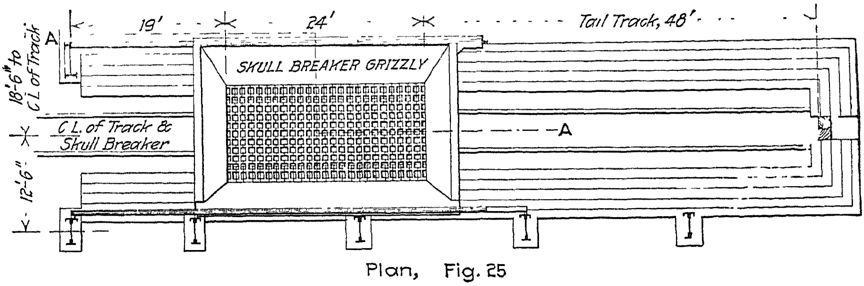


FIG. 25.—SKULL BREAKER, INTERNATIONAL SMELTING CO. PLAN.

CHIMNEYS

(a) *The Largest Steel Chimney in the World*

Fig. 26 shows a general elevation with details of the large steel chimney at the plant of the United Verde Copper Co. at Clarkdale, Ariz. It is 30 ft. 9½ in. in diameter inside the steel plates and is 400 ft. 1 in. in height, from the base to the top. It is believed to be the largest steel chimney in existence at this time. It has a brick lining 4 in. thick throughout its height.

(b) *Slag Foundation for Steel Chimney*

Fig. 27 shows the details of the foundation for a steel chimney at the smelting plant of the Calumet & Arizona Mining Co. at Douglas. The chimney is 305 ft. high from the top of the foundation and is 25 ft. 9½ in. in diameter, inside of the steel shell. It has a hollow-tile lining 4 in. thick throughout its height. The feature of the chimney is in the construction of its foundation. The foundation was cast from molten slag, hauled to the site for the chimney in the usual slag pots, instead of concrete or masonry as is usual for chimneys of this type. A template for holding the anchor bolts was made of structural angles and channels supported on a central concrete pier and held from turning or moving by a second concrete pier at the outer circumference. The foundation

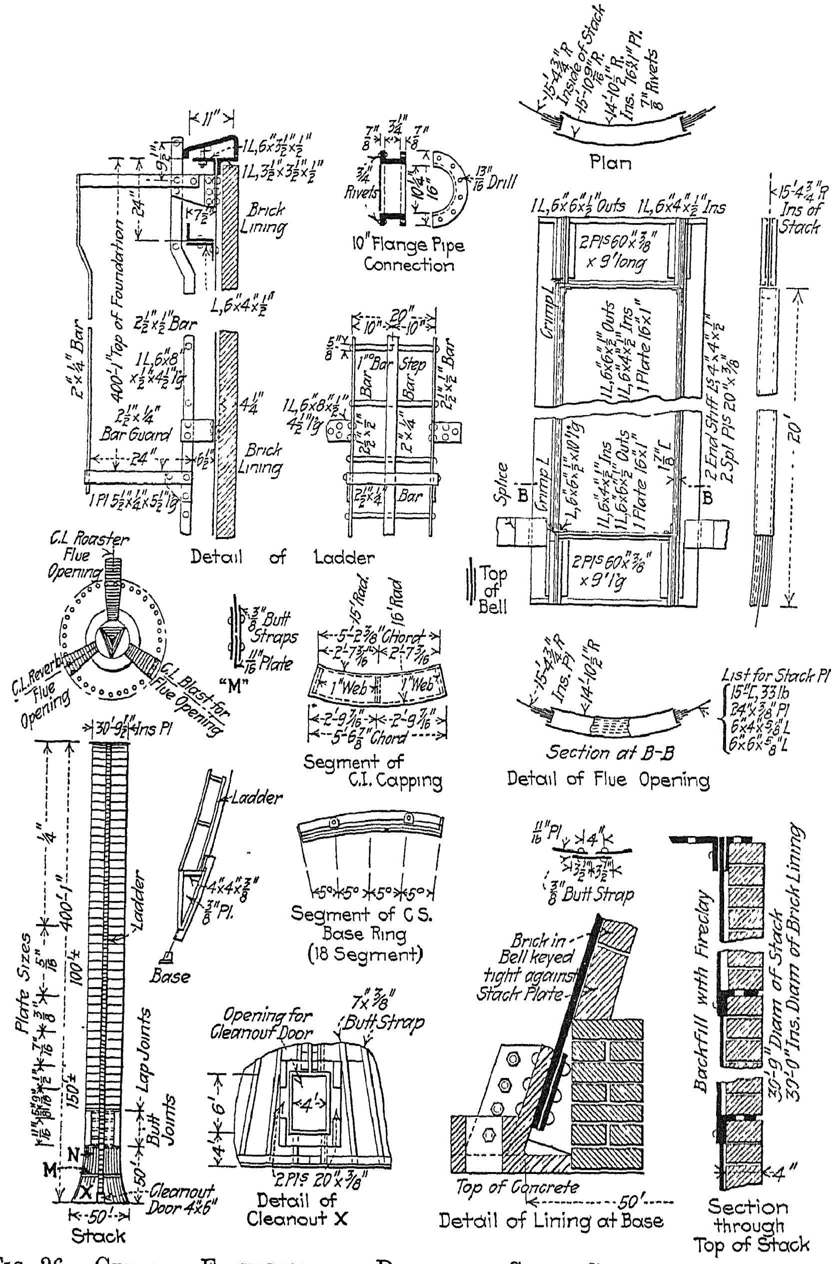


FIG. 26.—GENERAL ELEVATION AND DETAILS OF STEEL CHIMNEY, UNITED VERDE COPPER CO.

bolts were supported at their lower ends on small concrete piers and they in turn supported the template at other points of its outer circumference not supported by the concrete pier, each bolt having at the top a nut on the under side and one on upper side of the template,

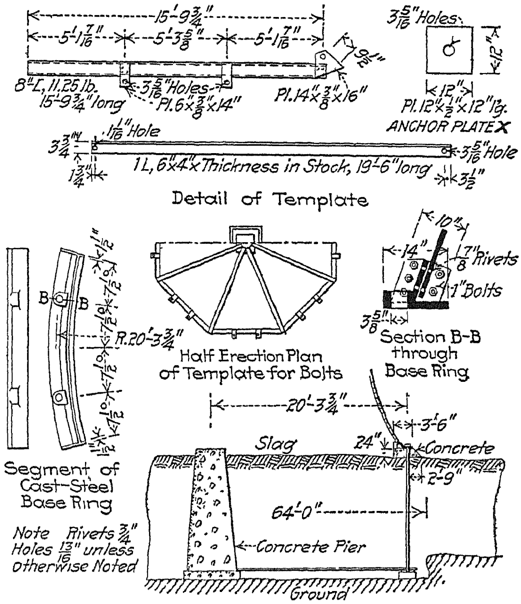


Fig 27.—Details of Slag Foundation for Chimney of Calumet & Arizona Mining Co., Douglas.

forming a secure support. Large washers were provided for the bottom ends of the anchor bolts over which were laid old steel rails. Blast-furnace slag was then poured over the foundation and adjacent ground, forming the foundation for the chimney. A concrete capping was laid on top of the slag for the cast-steel base-ring of the chimney proper.

Discussion

L. D. Ricketts, New York, N. Y.—The advance which has been made in smelting has been in the line of cheaper cost of handling, due to larger units and decrease in losses. At the International smelter, Mr. McGregor designed everything to handle the ore under cover. But we do find an unaccounted for loss of 0.7 per cent. of our copper, which is serious. Mr. Wallace has made a suggestion that I think metallurgists ought to know of and think about. He made it years ago—and that is that one of the chief losses is undoubtedly in the charging and discharging of the calcine cars. His idea is that possibly the dryers

and heaters might be put immediately over the reverberatories so that they could discharge in conveyors. They could feed on both sides, and thereby the dust raised by discharging in the cars from the hoppers could be overcome.

E. P. Mathewson, Anaconda, Mont.—I might say a few words on the plants we have visited so far. I was particularly struck with the cleanliness of the plants, which is next to Godliness. I noticed also that the skull-cracker arrangement they have for handling their converter slag is very good. And they have paid considerable attention to ventilation, and are taking steps to take care of the health of the men. "Safety first" has been well looked after in all the plants we have visited so far. The steel structures are right up to date, and the buildings are of the most modern type, and everything is conveniently arranged. I think the designers of the plants in the Southwest are to be congratulated on what they have done.

Smelting at the Arizona Copper Co.'s Works

BY F. N. FLYNN,* CLIFTON, ARIZ.

(Arizona Meeting, September, 1916)

Introductory

IN 1882, The Arizona Copper Co., Ltd., acquired producing copper mines at Metcalf and Morenci (locally called Longfellow). Metcalf is situated a distance of 7 miles, and Morenci a distance of 6 miles from the general office of the company at Clifton, Ariz. A reduction works with five blast furnaces was built in the center of the town of Clifton in 1884 and operated until January, 1914.

Concentration of oxidized ores was begun in 1890–1891, in a mill built adjoining the smelter, and in October, 1893, a leaching process was started for treating tailings from the oxide mill. A sulphide concentrator with 300 tons capacity was built at Clifton and started treating chalcocite ores in July, 1896. This was the first mill to treat low-grade copper-sulphide ores successfully. Another concentration mill for sulphide ores was erected at Morenci in 1900. The sulphide mills are now being remodeled to treat ores by concentration and flotation, with a capacity of 4,000 tons at Morenci and 500 tons at Clifton. The leaching plant has not been running since September, 1914.

The average yearly production from this company's mines and reduction works from 1885 to date has been:

	Pounds
10 years ended 1894	6,864,902
10 years ended 1904	20,439,614
10 years ended 1914	32,665,905
Part of year 1915, to Sept. 12	30,206,106
4 months of 1916, to June	15,964,840
June, 1916	4,900,000

Construction on a new smelting plant was started in February, 1912, and finished in February, 1914. In the *Transactions*, vol. 49, E. Horton Jones published "Unit Construction Costs from the New Smelter of the Arizona Copper Co., Ltd.," which fully describes the construction of this plant, with details of costs.

* Superintendent of Smelting Department, The Arizona Copper Co., Ltd.

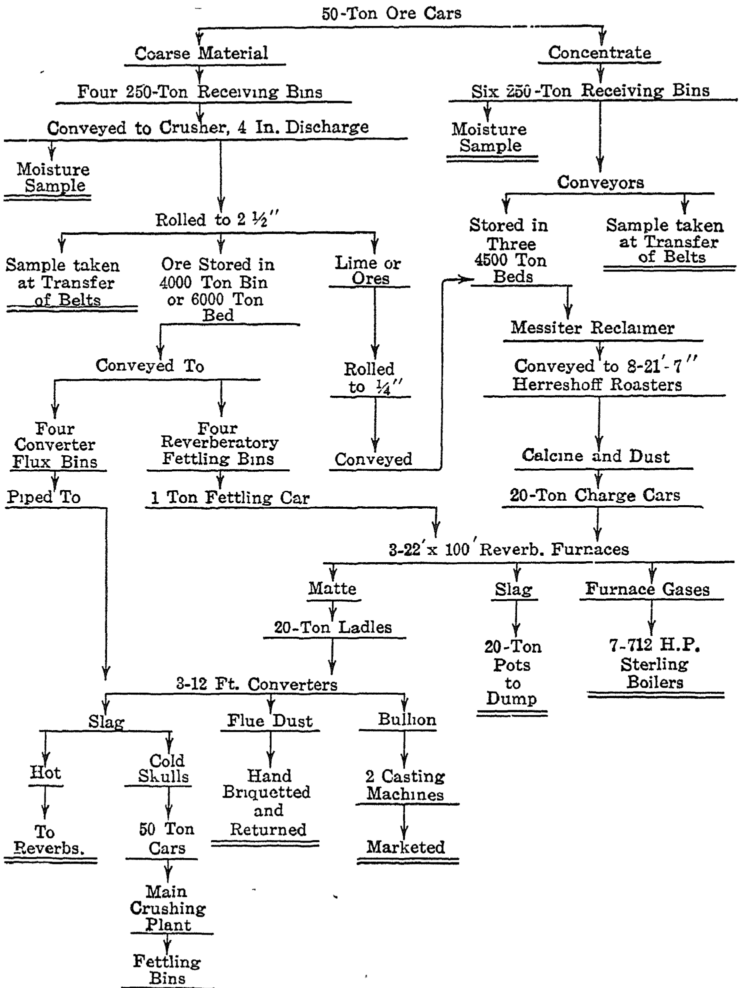
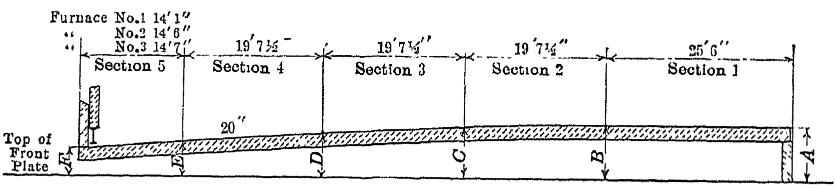


FIG. 1.—FLOW SHEET OF THE ARIZONA COPPER CO'S. SMELTING PLANT



	Reverb. No.1	No.2	No.3		Reverb. No.1	No.2	No.3
A	7'8⅝"	6'10⅛"	7'10"	*D*	6'4½"	6'7¾"	6'0¾"
B	7'10⅜"	6'11½"	7'4"	*E*	5'4"	5'6⅜"	5'6"
C	7'0⅜"	6'9⅝"	6'10⅝"	*F*	4'0⅛"	5'0½"	4'8⅜"

FIG. 2.—SKETCH SHOWING POSITION OF REVERBERATORY ARCHES ON CENTER LINE.

The new smelter site is on gently sloping ground, 2 miles below the town of Clifton, and close to the San Francisco River. The Arizona & New Mexico standard-gage railroad and the Coronado narrow-gage railway serve the plant. Smelter yard standard-gage trackage consists of 2.38 miles for steam locomotives, and 1.72 miles for electric haulage.

Reference to the accompanying flow sheet will assist in following description of smelting practice at the new smelter.

Roaster Division

Roaster equipment consists of eight Herreshoff furnaces with steel shells 21 ft. 7 in. in outside diameter. Hearths and linings in the different furnaces are built of materials as shown in the accompanying table.

Hearth and Lining Materials in Herreshoff Roasting Furnaces

Furnace No.	Drier	Hearth Numbers From Top Downward						Shell
		1	2	3	4	5	6	
1	Red brick	Firebrick	Firebrick	Gravel-concrete	Gravel-concrete	Gravel-concrete	2-in. concrete	Red brick
2	Red brick	Firebrick shapes	Slag-concrete	Firebrick shapes	Firebrick shapes	Firebrick shapes	2-in. concrete	Red brick
3	Red brick	Firebrick	Slag-concrete	Firebrick	Slag-concrete	Brick-concrete	2-in concrete	Red brick
4	Red brick	Firebrick	Firebrick	Gravel-concrete	Gravel-concrete	Gravel-concrete	2-in. concrete	Red brick
5	Red brick	Firebrick	Firebrick	Firebrick	Firebrick	Firebrick	2-in. concrete	Firebrick
6	Red brick	Firebrick	Firebrick	Firebrick	Slag-concrete	Slag-concrete	2-in. concrete	Red brick
7	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	2-in. concrete	Red brick
8	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	Firebrick blocks	2-in. concrete	Red brick

Hearths designated "gravel concrete" were made of river gravel, 3-in. size and under, 2 parts; clean sand, 4 parts; and cement, 1 part. Slag-concrete mixture consisted of crushed slag (all the size that passed a ½-in. screen) 3 parts, clean sand, 3 parts; and cement, 1 part. Composition of this slag was: SiO_2, 47.0 per cent.; Al_2O_3, 8.5 per cent.; Fe, 27.5 per cent.; CaO, 3 3 per cent.: S, 0.5 per cent. In the fifth hearth of No. 3 furnace, crushed red brick was substituted for gravel in the concrete mixture.

The first nine concrete hearths built were reinforced with iron rods laid in top and bottom courses. Reinforcing in those with inner drop holes weighed 3,093 lb., and those with outer drops 2,148 lb. When fully heated the fourth and fifth hearths, with heavy reinforcement, rose 6 in. at the center of the arch, which necessitated chipping off some of the concrete from the center of hearths. In later construction ½-in. diameter reinforcing rods were used. This reduced the weight of iron to 1,172 lb. and 1,373 lb. respectively for inner and outer drop hearths.

Slag-concrete hearths are preferred, providing the slag is highly

siliceous. All concrete hearths have given entire satisfaction. They have been in service since May 3, 1914, without any repairs.

Provision is made for cooling the central columns and rabble arms with air supplied from a motor-driven fan, but during the last 18 months cooling air has not been required.

The roaster charge has varied from ore, limerock and concentrate mixtures with 19.2 per cent. sulphur and 10.7 per cent. copper, to all concentrates with 30.7 per cent. sulphur and 15.8 per cent. copper, with fuel requirements varying from a maximum of 40 lb. of coal per ton of calcines to none when roasting all concentrate. The units of sulphur burned off have ranged from 7.5 to 17.4, the usual practice being to leave equal quantities of sulphur and copper in the calcine, because of the oxidized ore used in reverberatory fettling.

Normally, the material roasted requires only a little more heat than is derived from the sulphur burned. Heavy crude oil was first used as fuel, but with poor results, because with the low draft maintained burners could not be obtained of suitable size to burn continuously the small quantity of oil necessary to furnish this deficiency of heat, and distribute it properly over the hearth. The flame was admitted through one door on the third or fourth hearth from the top. A burner that would stay lighted in these comparatively cold hearths, produced an intense flame, and if kept running long enough to ignite the charge to the temperature desired, the brick hearth over the burner became too hot and charge fused on the hearth directly in front of the burners. Calcine produced was not uniform, because the sulphur had to be controlled by burning oil intermittently, which resulted in first a hot furnace and low sulphur, then a cold furnace with high sulphur.

Coal-fired Dutch-ovens with 12 sq. ft. of grate area have been adopted for supplying additional fuel required for roasting. Two ovens for each furnace admit the flame on the fifth hearth from the top. With these, the roast can be quickly and closely regulated, and the furnace repairs have been greatly reduced as compared with oil-burning conditions.

Ignition usually takes place on the fourth hearth from the top, and the temperature of the charge rises to 1,200 or 1,300° F. on the two lower hearths. With one reverberatory running, each roaster operated normally makes 65 tons of calcine per day. When two reverberatories are taking charge, each roaster has to supply 80 tons of calcine. Results show that while removing the same units of sulphur per ton, a furnace will consume less coal per ton when producing 60 than when producing 80 tons of calcine. Monthly averages of 113 tons of calcine produced per furnace day show a very high coal ratio. Furnace dampers are kept closed just to the point of smoking. Gases leaving the furnaces first enter a hopper-bottomed header-flue, through which they travel at a velocity of 6 or 7 ft. per second, at 325 to 425° F. tempera-

ture, and then pass into a large dust-chamber. Dust recovered from this chamber amounts to 0.8 per cent. of the dry charge to roasters.

The dust-chamber roof, made of No. 11 plate steel, shows slight deterioration at the laps. A sheet-copper cap laid over the joint at apex of roof, lasted 2 years and has been replaced by concrete.

Reverberatory Division

Smelting for production of matte is all done in reverberatories, either one or two furnaces being operated as required. Three furnaces set parallel to each other the long way, and 26 ft. apart, each with a hearth area 22 by 100 ft., are available, one of these always being a spare. The hearth in No. 3 furnace is crushed silica fused in on a slag bottom. Furnaces Nos. 1 and 2 have hearths of crushed silica fused in on bottoms of broken quartz. Roofs were all originally built with 1 in. to the foot rise in the arch. When the roof of No. 1 furnace was rebuilt, the arch was given a rise of 1¼ in. to the foot. Six-inch holes, spaced 24 in. centers, left in the roofs along the side walls and across the back, allow fettling material to be piped into the furnaces from trough-shaped bins built vertically over these holes. Five-inch diameter pipes on fettling bins are being changed to 8-in., to permit using coarser material without choking the pipe openings. Other dimensions of these reverberatories that affect their operation are given in an accompanying table.

Reverberatory Dimensions

	Furnace Number		
	1	2	3
Lowest matte tap hole to top of front plate...	1 ft. 10⅝ in.	.....	1 ft. 3⅞ in.
Matte tap hole to skimming block, in use at present..................................	10⅞ in.	.....	1 ft. 1 in.
Height of oil burners above front plate......	1 ft. 11⅛ in.	.....	1 ft. 8⅜ in
Height of oil burners above skim block at present in use..........................	1 ft. 11⅛ in.	.....	1 ft. 5⅞ in
Furnace throat area when slag line is level with top of front plate, square feet..............	17.4	26.2	22 0
Furnace throat area, when slag line is raised 6 in. above top of front plate, square feet...	13.2	22.6	18 0
Area of opening in verb shaft, square feet....	40 0	28 5	28.5
Area of intake opening to header flue, square feet...	45.6	55.2	54 6
Thickness of side walls, as rebuilt, above matte line, inches....	30.0	30.0	12.0

NOTE.—The 12-in. side wall is more favorable for greater height of fettling.

In furnaces Nos. 2 and 3 the weight of the shaft has been taken off of the verb arch, by means of a telescopic shaft supported on steelwork. Fig. 2 shows the relative positions of the reverberatory arches on the center line of roof. This height is satisfactory when burning small quantities of oil, but might be raised considerably in Sections No. 1 and 2 for larger quantities of oil.

The following log of the third campaign of No. 3 reverberatory furnace is representative of operations on a one-furnace schedule:

Furnace started May 20, 1914.

6.208 days lost in August, repairing side walls, verb shaft, and 24 ft. of new roof on Section 1.

0.769 days lost in September, October, November and December, 1914, making minor repairs.

1.122 days lost in January, 1915, patching roof and front end.

2.736 days lost in March, 1915. New roof on Section 1.

5.642 days lost in July, 1915. 50 ft. roof repairs on Sections 1 and 2, and new jambs under header-flue arch.

Furnace stopped Sept. 12, 1915, on account of labor strike, but was in good repair at that time.

16 477 days lost time in 16 months total campaign.

Dry Tons Solid Charge Smelted:

During total campaign	178,165.0
Daily average for total campaign	384.6
Daily average for one month, highest	512.4
Daily average for one month, lowest	312.2
Highest for one day, during highest average month	543.0
Lowest for one day, during highest average month	413.0

Barrels of Oil per Ton of Solid Charge:

Lowest month	0.653
Highest month	0.951
Average during campaign	0.744

By a campaign is meant the elapsed time between starting a furnace and tapping out the matte. New roofs on Sections 1 and 2 were built upon ore centers, without tapping the matte. The last side walls repaired were in November, 1914. Since adopting greater fettling height, no further wall repairs are anticipated.

Two parallel charge tracks cross the furnaces near the back in a direction at right angles to their length. Fifteen-ton charges of calcine, at a temperature of 1,050 to 1,150° F., are dropped through one water-jacketed charge hole in the center of the roof on either track, but 90 per cent. of all charges are dropped from the back track. This part of the charge dropped through the roof is designated "direct charge." Seventy-five per cent. of the total solid charge smelted is introduced in this way.

Bad furnace conditions are always expected if such cold material as crude ore, slag or limerock of any size, or anything coarse like siliceous ore agglomerated with converter slag, either cold or warm, is dropped as direct charge without mixing it well with hot calcine. Ore, limerock or clinker produces "charge floaters." Cold slag sinks and makes sticky furnace bottoms. When it is necessary to add cold or coarse material to the direct charge, the best results are obtained when they are well mixed with hot calcines, in the calcine charge cars, but better still in the roasting furnaces.

Bed mixtures to be roasted are made of the required composition to produce a slag, when smelted, in which the ratio of oxygen in acids to oxygen in bases will fall between 2.00 and 2.25. A vast difference, however, is found in the smelting properties of calcines made from these mixtures, regardless of the similarity between their analyses or the analyses of the slags they produce. Calcine made from concentrate or sulphide ores of ½ in. size smelts readily, whereas if the calcine is an artificial mixture made of siliceous oxidized ore crushed to this size, and mixed with an excessive proportion of limerock, the charges melt more slowly. The results from the two following mixtures will fairly illustrate this point:

Class of Material	Mix 103. Per Cent. of Total	Mix 120. Per Cent. of Total
Sulphide concentrate	100.0	70.7
Siliceous ore, low sulphur	.....	4 8
Siliceous oxide ore	.....	12.9
Limerock	.....	11.6
Total	100.0	100.0

Mix No.	Analysis						
	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Fe, Per Cent.	CaO, Per Cent.	S, Per Cent.	Cu, Per Cent.	Oxygen Ratio
103	18.3	4.6	28.2	0.7	29.0	14.10	1.43
120	20.7	4.7	24.1	7.6	22.5	10.98	1.45

A daily average of 365 tons of calcine per furnace was smelted during the time that mix 103 was on the charge. In comparison, 200 tons per furnace day was barely maintained while the charge consisted of calcine from mix 120. The practice is to crush siliceous ore as fine as 3/16-in. for roaster beds, and endeavor to keep the sulphide material as high a

75 per cent. of the total bed mixture. Ore and limerock mixtures require a much higher oil ratio than calcine from straight concentrate.

In other words, the greatest percentage of oxidized material which it is permissible to smelt in the reverberatory furnace is found to be:

	Tons per Furnace Day	Per Cent.	
Direct charge	300	25	Oxidized ore and limerock.
Fettling	100	100	Oxidized ore.
Total	400	44	Oxidized ore equivalent.

Above 44 per cent. oxidized ore, the charge could undoubtedly be better handled in a blast furnace. The charges spread better when the furnace is nearly full of matte, and also when the copper and sulphur in the calcine are equal. Matte under 40 per cent. copper is favorable for higher tonnage.

Fettling.—Ore and byproducts crushed to a 2½-in. size for fettling constitute the remaining 25 per cent. of the solid charge smelted. It is customary after each skim to feed these materials from bridge-wall to shaft in the following proportions:

Furnace	Section No.	Per Cent.	Material
Back	1	25	Old smelter slag. Converter slag skulls.
Back	2	50	Less siliceous ore.
Middle	3	12	Less siliceous ore.
Middle	4	8	Less siliceous ore.
Front	5	4	More siliceous ore.
Shaft	..	1	Silica.

	Analysis							Oxygen Ratio
	SiO_2, Per Cent.	Al_2O_3 Per Cent.	Fe, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	S, Per Cent	Cu, Per Cent.	
Converter slag skulls	19.6	3.9	47.2	0.9	0.0	3.6	9.00	0.89
Old smelter slag	37.2	8.9	22 0	10.3	5.2	0.3	3.56	2 12
Less siliceous ore	42 6	7.7	18 0	3.0	2.5	1.0	6 80	3.74
More siliceous ore	65.0	12.2	5.6	0.8	0.6	1 9	3.78	19.43
Silica	84.4	6.3	3 1	0.5	0.0	2.6	0.74	46.34
Approximate average	40.4	7.5	21.5	3.5	2.4	1.3	6.49	3.08

Prior to July, 1914, the height of fettling was carried not to exceed 6 in. above the slag line. This height was gradually increased until October, 1914, when the practice of fettling to the holes in the roof was commenced and later successfully adopted after the size of fettling material was increased from 3/4 in. to the present 2 1/2-in. size. The coarse size stands better. It reposes at an angle of 45° above and 45° to 60° below the slag line.

The results achieved in performing fettling operations as practised here, depend upon the relative location and manner in which the material is dropped, as well as the amount of fines in it and its chemical and physical composition, as may be concluded from the following account of experience with different fettling materials.

1. Slag and byproducts crushed fine work unsatisfactorily in any part of the furnace. However, if crushed coarse, results are good when such material is charged in the back of the furnace.

2. Less siliceous fine ore is not good fettling in any place in the furnace. It does not stand, and requires care to keep it from running out into the furnace. Crushed coarse, it makes good fettling wherever used.

3. More siliceous fine ore mixed with coarse byproduct make a bad combination. The only place the more siliceous fine ore alone can be used with any degree of success is on the bridge-wall.

4. More siliceous coarse ore is not so acceptable as less siliceous coarse ore, except in the front half of the furnace, because it makes too much siliceous scum on top of the slag.

5. Coarse raw concentrate is good in the back half of the furnace, but does not stand so well nor so high as coarser material. It gives bad results in the front half because it melts easily to a magnetic mush.

6. Fettling with hot calcine can only be done with any degree of success in Section No. 1, and then only slightly above the slag line. At that elevation it runs out over the furnace bottom. An attempt was made to fettle the entire furnace with calcine, but was abandoned when the tonnage dropped to less than half capacity. Calcine gave poorer results than any other material experimented with.

7. Pyrite in the front half melts easily to a magnetic mush. In the back half when mixed with siliceous oxide ore it works well.

8. The ideal fettling material was found to be less siliceous copper-sulphide ore containing 5 to 10 per cent. sulphur.

In other words, basic or sulphide materials are charged near the firing end because of their lower fusing points. If charged in the front portion, where the temperature is lower, they melt with sufficient ease to make a thick, mushy, semifused slag, and in the case of pyrite make quantities of magnetite. Siliceous ores with 60 per cent. silica produce a scum which blankets the slag and causes higher oil ratio and lower tonnage. If the siliceous ores contain much copper in oxidized form,

the scum will assay from 0.7 to 1.3 per cent. copper, whereas if the copper is present as a sulphide the scum will assay very much lower.

The most important development as regards fettling has been along the line of coarser crushing and greater height of material to protect the side walls without encroaching on the furnace bottom area. Fettling "floaters" are unknown. The successful results of this practice are best seen from the photograph of one of the furnaces (Fig. 3).

Fuel is 14 Bé., 18,000 B.t.u., California crude oil, pumped at 120 lb. pressure and delivered through steam heaters at a temperature of 170° F. to six burners and atomized with converter air. Two hundred

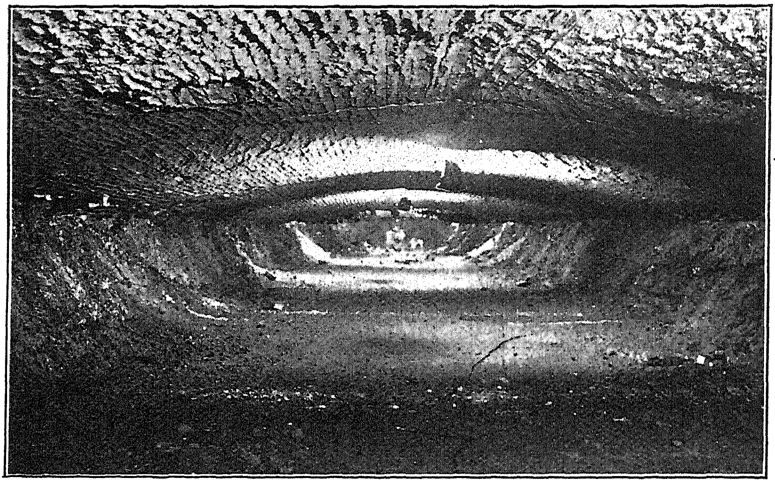


Fig. 3.—Reverberatory Furnace No. 3 of Arizona Copper Co., Ltd.

Campaign No. 3. Started May 20, 1914, finished Sept. 12, 1915 (date of labor strike.)

Dry tons of solid charge smelted: total 178,165; highest daily average for one month, 512.4.

Fettled to the roof with "less siliceous" oxide ore and converter slag skulls, crushed to 1½ in. Slope of fettling, from 45 to 60°. Fettling material amounts to 25 per cent. of total solid charge.

and forty to 365 bbl. of oil is burned per furnace-day. This variation covers a one-furnace schedule when hard firing for high tonnage is required, or two furnaces with light firing and correspondingly low tonnage smelted.

Furnace-throat draft (inside the furnace) is 0.12 to 0.16 in. of water, and in header-flue common to all furnaces, draft is 0.50 in. water. Gases leave the furnaces at temperatures from 1,800 to 2,000° F. Seven Stirling boilers are available for reverberatory waste heat, each 712-hp. capacity, and all connected in multiple through cross-over flues from the header-flue. Dampers are so arranged that gases can be sent to any one or all of the boilers. There is no bypass to the chimney. Four or five boilers

are used when running one furnace. Reverberatory dust caught in all the boilers does not exceed 5 tons monthly when running one reverberatory.

Furnace gases leaving the waste-heat boilers at temperatures varying between 471° and 588° F., are conducted through 6-ft. diameter steel pipes into a flue connecting with the main chimney. The temperature of gases in the flue ranges between 400° and 500° F. at different points, and at these temperatures the average velocity of gases is 10 ft. per second, when running one furnace. An average analysis of this gas is SO_2, 0.3 per cent.; CO_2, 5.9 per cent.; O, 12.9 per cent.; CO, 0.4 per cent.; N, 80.5 per cent.; SO_3, not determined. When the flue was cleaned recently, 100 tons of dust was removed that had accumulated during 2 years' furnace operations. The composition of this dust was:

	SiO_2	Al_2O_3	Total Fe	Ferrous Fe	CaO	S	Cu	Free H_2SO_4	H_2O
Reverberatory chamber dust	17.4	6.3	11.8	...	2 3	10.3	9.65	3.0	9.5
Soluble in water	....	1.8	3.7	2.6	2 1	9.1	6.60		
Per cent. of total, soluble in water	. ..	28.6	31.4	....	91.4	88.3	68.4		

The flue roof was originally made of No. 14 steel plate. This had to be repaired after 8 months' service. Holes were eaten in the plate and in spots the whole sheet was pitted and nearly eaten through. Moist flue dust covered the inside surface of the roof. This deposit consisted principally of sulphates of iron and copper, as is shown by the following composition calculated from analysis of dry sample:

Insoluble	3.0
$CuSO_4$	7.7
$Al_2(SO_4)_3$	8.0
$FeSO_4$	18.7
$Fe_2(SO_4)_3$	45.3
$ZnSO_4$	3.2
$CaSO_4$	2 9
Free H_2SO_4	12.3
Total	101.1

This would not jar loose from the sheet, and metallic copper had deposited on the iron. Concrete slabs reinforced with wire netting were laid on top of the steel plate and the joints between slabs filled with asphalt. The concrete covering has served 2 years. The corrosive action of gases on the iron was arrested for 10 months of operations after the slab roof

was put on, but during an idleness of 5 months, due to labor trouble, the plate had corroded completely.

Lime in the mortar used in laying tile walls of this flue was partly changed to sulphate by the furnace gases. While the flue was cold, the walls swelled far out of plumb. The tile broke and walls had to be rebuilt. New walls are brick laid in slime mortar.

Converter slag is poured through the back wall of furnace, at the center, through a launder discharging at the height of the oil burners. Magnetite builds up under the discharge of the launder, but the bridge-wall on either side of it requires fettling.

Considerable has been written regarding the fluxing value of converter slag poured into reverberatory furnaces. The prevailing opinion seems to be that very little fluxing value is within one's control. It is argued that the basic converter slag does not act upon the charge resting on the slag surface, does not mix with the trisilicate slag, and passes through the furnace like a submarine as a monosilicate, emerging near the skimming end. The technical reason advanced for these arguments is the relatively high specific gravity of the basic converter slag. The writer has proved to his own satisfaction that practically all of the iron present in liquid converter slag, other than iron in the magnetic state, is effective as flux. The violent boiling and the reactions that take place when calcine charge is dropped through the central charge hopper are effective in mixing any slag that may have separated into layers in the furnace at the back. If it is true that the charge dropped in the back of the furnace partly melts to a trisilicate slag, underlaid by the bisilicate, which in turn rests upon the monosilicate converter slag, it would be a difficult task to explain why the surface or trisilicate slag is so active in its consumption of fettling material at the slag line. On the other hand, it is not difficult to imagine the monosilicate slag attacking the fettling material along the side walls, from the matte line up. How it would fail to do so would be difficult to explain.

The effect of liquid converter slag was vividly shown on one occasion, when a furnace had been tamped with moist quartz from floor to roof along the walls, and when fused was largely filled with hot converter slag and little regular charge. Within 24 hours there was little quartz remaining.

On several occasions, two furnaces running on identical charges have been used for illustrative purposes. Into one, all of the converter slag was poured. In the other, without converter slag, the full quantity of limerock equivalent to the total fluxing value of the converter slag was required.

There are others who contend that even magnetite in converter slag is acted upon by the furnace matte and becomes effective as flux. Our own experience is that whenever magnetite accumulates in the furnace,

it must be pulled out with the rabble as mush, the necessary limerock added to the charge, and plenty of sharp drills made ready at the matte tap hole.

The absence of an accurate method for the determination of magnetic oxide of iron in furnace byproducts is one of the most unfortunate and regrettable conditions with which the metallurgist is confronted, and renders the elucidation of this obscure subject difficult. Without some means of quantitatively determining this substance, accurate data as to its fluxing value can not be obtained from balance sheets even if materials and products are all weighed, because it is not certain from an analysis that the magnetic iron has been changed in passing through the furnace.

Converter Division

The production of magnetite in the converters is, within reasonable limits, controlled by the converting practice. The converter slag poured into the reverberatory furnace should not contain any magnetite visible to the eye. The safest practice is to reject the last 2 cu. ft. of slag when pouring. The rejected slag with the ladle skull will eventually enter the furnace as fettling material, where its detrimental effect to the furnace is temporarily turned to some good, because of its refractory properties. Its tendency to stick to the sides when mixed with cold ore is quite noticeable when the fettling bank is removed from the walls when the furnace is out of service. By this practice, magnetite is made to perform a somewhat similar service in the reverberatory furnace to that which it performs in the protective coating in a basic converter. Unfortunately only a small portion of the total magnetite production can be so used.

Some metallurgists prefer to make a small quantity of magnetite on each converter charge, contending that in that way the coating remains a more constant thickness. They accomplish this by blowing a few minutes on the start or finish, either or both, but usually at the start, with a shortage of silica. The writer is of the opinion that by intentionally making magnetite with every charge, the quantity made must be very large as compared with the small amount which can be made to stick to the walls by so haphazard a practice going on continuously. Our practice is to coat the shells when the lines of brickwork can be seen through the thin coating. And, when a shell is being coated, no attempt is made to do anything with the matte other than form the coating. When coated, the magnetite and copper mush is dumped on the floor under the shell; later it is charged in small quantities at intervals in other charges. When coated, a shell is never blown without a large excess of silica. Although siliceous ore is used for flux, 350 lb. of 80 per cent. silica is used to granulate the final slag on the copper finish, to insure the removal of the last traces of iron from the copper with the more active silica. Since

this quartz, with any magnetite which it has collected, remains in the converter for the next charge, the magnetite is in contact with the quartz particles from the start of the blow.

The converting division is equipped with five 12-ft. Great Falls-type shells lined with magnesite brick; three converter stands, each operated by an independent motor (the fourth stand is under construction); two 40-ton cranes; two straight-line casting machines, each with 39 copper molds. Two stands are regularly used, for which air at 13 lb. pressure is supplied from power house by two of three Nordberg blowing engines.

When converting matte of over 40 per cent. copper, only a small amount of cold matte shells and cleanings from the floor can be used; but with lower-grade matte, all matte shells, converter flue dust, and any converter byproducts made except converter-slag skulls can be handled in this division.

Molten copper is transferred from converters to casting machines in cast-steel ladles lined with fines screened from ores regularly used as converter fluxes. Bars are cast in molds made of converter copper. A 1¼-in. cast-iron splash plate covers half of the bottom area, and an average of 73 tons of bullion is cast per mold. Bars weigh 240 lb. each and 35 min. is required for casting a charge weighing 7 tons. Considerable chipping of bullion bars is necessary to remove edges and fine shot due to blowing to gas finish of 99.60 per cent. copper.

Comparison of One-Month Periods with One and Two Reverberatory Furnaces in Operation

Roaster Division	One Reverberatory	Two Reverberatories
Total tons dry charge roasted	14,257.0	17,235 0
Dry tons charge per furnace-day running time	78.4	73.9
Analysis of charge:		
SiO_2, per cent	17.3	19.8
Al_2O_3, per cent	4.7	4.5
Fe, per cent	28.2	24.3
CaO, per cent	1.6	6.4
S, per cent	28.8	24.5
Cu, per cent	13.61	12.08
Oxygen ratio	1.33	1.44
Analysis of calcine:		
SiO_2, per cent	20.0	22.0
S, per cent	13.8	13.8
Cu, per cent	15.70	13.44
Weight of calcine produced as per cent. of dry charge	86.7	89.9
Units sulphur eliminated	15.0	10.7
Pounds coal used per ton of roaster charge	4.2	14.2
Waste-gas temperature, degrees Fahrenheit	428.0	396.0

Reverberatory Division	One Reverberatory	Two Reverberatories
Dry tons direct charge per furnace-day	402 5	262.7
Dry tons fettling per furnace-day (ores)	84.9	65.7
Dry tons fettling per furnace-day (byproduct)	25.0	38.7
Dry tons total solid charge smelted per furnace-day	512.4	367 1
Per cent. of fettling to total solid charge	21.45	28 3
Analysis of direct charge:		
SiO_2, per cent.	19.8	21.7
Al_2O_3, per cent.	5.4	4 9
Fe, per cent.	32.3	26 6
CaO, per cent.	2 3	7.7
S, per cent.	13.7	13 6
Cu, per cent.	15.55	13.24
Oxygen ratio	1.32	1.41
Analysis of fettling (ores and byproducts):		
SiO_2, per cent.	38.8	39.5
Al_2O_3, per cent.	7 6	8.0
Fe, per cent.	23 4	25 5
CaO, per cent.	1 7	4 0
S, per cent.	9.1	3 0
Cu, per cent.	6 65	5 52
Oxygen ratio	3.36	2.93
Total tons hot converter slag to reverberatories	2,419.0	2,862.0
Analysis of reverberatory slag:		
SiO_2, per cent.	35.6	38 0
Al_2O_3, per cent.	9.7	9.0
FeO, per cent.	49.7	39 4
CaO, per cent.	2.4	8.4
MgO, per cent.	0.6	1.5
Cu, per cent.	0.51	0.45
Oxygen ratio	2.04	2.06
Analysis of matte, per cent. Cu	37.43	37.70
Per cent. matte fall	31.68	27.1
Per cent. sulphur volatilized in furnace	24.14	25.7
Bbl. oil burned per ton solid charge smelted	0.653	0.778
Bbl. oil burned chargeable to steam	0.315	0.349
Bbl. oil burned account smelting	0.338	0.429

The accompanying 2 months' results were selected for comparison, because they show the extreme variation in the material treated in this department. Limited storage capacity at the smelter required two furnaces to be operated a part of the time to handle receipts. Additional charge needed for the month that two reverberatories were operated was composed of siliceous oxidized ores and limerock, and represents a temporary condition. The concentrator additions, now almost finished, wil in the future supply sufficient concentrates to make the proportions o

total material treated fall somewhere within the percentage limits given in the examples.

Converter Division	One Reverberatory	Two Reverberatories
Tons reverberatory net matte treated	5,293.0	5,903.0
Per cent. Cu in reverberatory net matte treated	37.43	37.70
Tons siliceous ore used for flux	820.0	1,356.0
Analysis of siliceous ore used:		
SiO_2, per cent	64.5	65.3
Al_2O_3, per cent	10.6	11 6
Fe, per cent	5 2	5.7
CaO, per cent	0 2	0.5
S, per cent	0.5	1.7
Cu, per cent	6 45	4.29
Oxygen ratio	25 30	22 65
Analysis of converter slag produced, per cent. SiO_2	19 6	19.6
Tons copper bullion produced	1,800 0	2,450 0
Cu in bullion, per cent	99 60	99.55
Tons copper produced per converter-day running time	43 9	51.4
Minutes blowing per ton copper	33.0	28 0
Average tons bullion per charge	6 429	8 524
Average time of blowing a charge	3 hr. 54 min.	3 hr. 59 min.
Tons cold new material treated per ton bullion	0.462	0.592
Tons cold byproduct re-treated at converters per ton bullion	0.849	0 577
Blast, pounds pressure	13 0	13.0
Cu. ft. air per converter per minute	6,738 0	5,445.0
Cu. ft. air per ton bullion produced	220,949 0	152,499.0
Cu. ft. air per ton iron and sulphur eliminated	135,043 0	96,145.0
Oxygen efficiency, calculated from blower displacement and requirements of Fe and S eliminated	60.63 per cent.	84.09 per cent.
Total Material Treated—All Divisions:		
Dry tons concentrates	13,227 0	13,360 0
Dry tons ores	4,069.0	7,303.0
Dry tons limerock	391.0	2,130.0
Dry tons new material, total	17,687.0	22,793.0
Dry tons byproducts treated at reverberatories	924.0	*2,317.0
Concentrates, per cent. of total new material treated	74.78	58 62
Ores, per cent. of total new material treated	23.01	32.04
Limerock, per cent. of total new material treated	2.21	9.34

* Including reverb. slag skulls

Smelter Power

Divisions	Kilowatt-hours One Furnace	Kilowatt-hours Two Furnaces	Kilowatt-hours per Ton New Material One Furnace	Kilowatt-hours per Ton New Material Two Furnaces
Sampling:				
Receiving, high-pressure air converted to kw.-hr.	1,813	2,590	0.10	0.11
Crushing, electric	10,501	21,548	0.59	0.95
Sampling, electric	1,146	2,351	0.06	0.10
Bedding, electric	3,819	7,836	0.22	0.34
Reclaiming, electric	22,576	22,843	1.28	1.00
Total	39,855	57,168	2.25	2.50
Roasting:				
Roaster furnaces, electric	7,692	7,495	0.43	0.33
Calcine cars, electric	3,140	5,093	0.18	0.22
Total	10,832	12,588	0.61	0.55
Reverberatories:				
Reverb. furnaces, electric	3,013	3,303	0.17	0.14
Reverb. furnaces, high-pressure air converted to kw.-hr.	453	647	0.03	0.03
Reverb. furnaces, low-pressure air converted to kw.-hr.	37,339	72,774	2.11	3.19
Slag railway, electric	4,420	10,451	0.25	0.46
Boilers, high-pressure air converted to kw.-hr.	1,813	2,590	0.10	0.11
Boiler feed-water pumps, electric	31,220	20,553	1.77	0.90
Oil pumps, electric	4,030	2,009	0.23	0.09
Total	82,288	112,327	4.66	4.92
Converting:				
Converters, electric	5,263	5,674	0.30	0.25
Converters, low-pressure air converted to kw.-hr.	218,235	206,802	12.34	9.07
Cranes, electric	22,790	19,635	1.29	0.86
Bullion casting machines, electric	2,340	2,920	0.13	0.13
Total	248,628	235,031	14.06	10.31
General Works:				
River well, electric	6,760	8,001	0.38	0.35
Boiler and machine shops, electric	2,900	2,730	0.16	0.12
Boiler and machine shops, high-pressure air converted to kw.-hr.	4,986	7,122	0.28	0.31
Yards, electric	1,313	1,439	0.07	0.06
Laboratory, electric	2,807	3,077	0.16	0.13
Total	18,766	22,369	1.05	0.97
Total, all divisions	400,369	439,483	22.63	19.25

Boiler Plant

There are 10 Stirling boilers, three 384-hp. direct-fired, and seven 712-hp. waste-heat. Water is measured in total and also separately to the two sets of boilers. Oil is measured to the direct-fired boilers. All boilers contain Foster superheaters, and deliver steam to the mains at 175 lb. pressure. The temperature of steam delivered at the power house is about 475° F. The temperature of the waste gases leaving the boilers varies between 471° to 588° F., depending upon the furnace practice and the number of boilers in service. The gases from one furnace practice are passed through five boilers. When two furnaces are in commission, six or seven boilers are used.

When smelting in one furnace, three direct-fired boilers are in continuous service to supply the power requirements, but the night load is light, therefore the boiler efficiency is low. The pounds of water evaporated per pound of oil averaged 11.98 over a period of 8 months. The oil averages around 15° Bé. and 18,200 B.t.u.

With two furnaces in operation, there is sufficient waste heat to generate all steam requirements for the night load, but oil must be used to carry the day load. Under these conditions, rather than keep the oil-fired boilers under steam over night, the three oil-fired boilers are not used. Instead, oil is burned under the waste-heat boilers. The oil so burned is measured and credited with evaporating 11.98 lb. of water, in the same manner as under oil-fired boilers on a one-furnace basis.

The net steam delivered to the power-house engines is credited to the boiler plant. Steam used in heating and atomizing oil and for other miscellaneous purposes, as well as steam wasted, is absorbed in the furnace-oil costs.

Under one-furnace conditions, the waste-heat boilers averaged over a period of 8 months, 7.71 lb. of water evaporated per pound of oil burned in the furnace.

The furnace-boiler division receives credit from the power house varying from 36 to 56 per cent. of the oil burned in the smelting furnace.

The indicated horsepower from the waste-heat boilers, per ton of solid charge smelted on a one-furnace basis over a period of 9 months averaged 98.71 i.hp.-hr. whereas on a two-furnace basis, over a period of 7 months, the average was 114.34 i.hp.-hr. from waste heat only.

The Basic-Lined Converter in the Southwest

BY L. O. HOWARD, GLOBE, ARIZ.

(Arizona Meeting, September, 1916)

WHAT was perhaps the first attempt at basic converting in the Southwest was made by the late Charles F. Shelby at Cananea early in 1907, when he removed the acid lining from one of the 8 by 12-ft. barrel-type converters then in use at the reduction works of the Cananea Consolidated Copper Co. and substituted a lining of one course of magnesite brick. In this shell, he blew 21 taps (about 180 tons) of matte to white metal before the brick gave out along the tuyère line. In every case, the white metal so made was transferred and blown to copper in other converters. The cost of the lining was $700 and Mr. Shelby gave up the experiment as impracticable. No further serious effort to bessemerize copper mattes in a vessel having a more or less permanent lining of basic brick was made by any of the large copper smelters in the Southwest until some months after Messrs. Peirce and Smith had brought the question permanently before the public, and had proved it to be a metallurgical and financial success by their work at Baltimore and Garfield.

Early in 1911, the Cananea Consolidated Copper Co. had in operation at its reduction works at Cananea, Mex., several 8 by 12-ft. barrel-type converters lined with magnesite brick which had previously been operating with acid linings. In the 18 months following, these converters produced 50,000 tons of copper. In June, 1912, this company installed the first of its new equipment of six stands of electrically operated 12-ft. Great Falls type converters, and shortly afterward discontinued the use of the smaller shells.

In September, 1911, the Consolidated Kansas City Smelting & Refining Co. had installed at its El Paso plant two 10 by 26-ft. Peirce-Smith converters. Later, in March, 1913, one standard Great Falls type converter was added.

About the time the Peirce-Smith converters were going in at El Paso, the Calumet & Arizona Mining Co. was experimenting with basic linings in 7-ft. by 10-ft. 6-in. barrel converters at its plant in Douglas, Ariz. These efforts were sufficiently successful to enable the plant to handle the total output of copper in these small shells until the standard Great Falls converters were substituted in June, 1914.

During the year 1911, the Anaconda Copper Mining Co., at its Great Falls plant, proved the worth of the so-called Great Falls converter. It was to this type, now being recognized as standard, that the Copper Queen Consolidated Mining Co. turned when it was decided to equip the reduction works at Douglas with basic-lined converters. In April, 1912, the first of these shells was put in operation. A second was added during the following month, and these two were quickly followed by six others.

In the meantime, the Consolidated Kansas City Smelting & Refining Co. had in operation, at its Hayden, Ariz., plant, two 10 by 26-ft. Peirce-Smith converters and later, in July, 1913, added one standard Great Falls converter.

The Arizona Copper Co. first used the basic-lined converter in October, 1911, at its old plant at Clifton, Ariz. The acid lining in some of the small converters was removed, and one of magnesite brick was substituted. The use of these small converters was continued until October, 1913, when the new plant with its equipment of three stands of electrically operated standard Great Falls converters was put in operation.

By January, 1913, the Old Dominion Copper Mining & Smelting Co. at Globe, Ariz., had installed a 12-ft. Great Falls type converter. It was the intention to handle in this shell the total output of copper, amounting to possibly 36,000,000 lb. yearly, but later, to insure sufficient converting capacity in case of accident to the larger shell, two of the old 7-ft. by 10-ft. 6-in. acid converters were lined with magnesite brick. It was not the intention, nor has it been the practice, to use these small converters unless absolutely necessary, and to date they have together turned out somewhat less than 1,500,000 lb. of copper.

The Detroit Copper Mining Co. at Morenci, Ariz., with a yearly copper output of about 11,000 tons, has never considered it advisable to adopt the large basic-lined converter. Instead, the 7-ft. by 10-ft. 6-in. acid-lined shells have been relined with magnesite brick and with these some remarkable work has been done, despite the serious difficulties that accompany the bessemerizing of copper mattes in such small basic-lined vessels. A record of over 3,100,000 lb. of copper converted with one lining, before any patching of the brick was necessary, has been made.

The Consolidated Arizona Smelting Co., at Humboldt, Ariz., likewise has continued in the very successful use of the small basic-lined converter, but contemplates the purchase of a large unit in the near future.

The International Smelting Co. at Inspiration, and the United Verde Copper Co. at Clarkdale, Ariz., have installed the standard Great Falls electrically operated converters in their new plants, both of which were put in operation during 1915. Unfortunately, owing to the newness of the latter plant and the fact that a proper working basis had not yet been

TABLE 1.—*Operating Data of Basic-Lined Converters in Southwest*

	1	2	3	4	5	6	7	8	9	10
Blast pressure in pounds	13	13.2	14	12	12.4	12–15	10–14	14	11.3	13
Cu. ft. air per minute per converter	5,445	7,691	10,600	4,283	6,054	7,250	......	8,181	4,033	4,117
Cu. ft. air per ton bullion produced	152,449	178,411	141,000	228,116	271,577	137,582	118,328	225,674	245,975	370,566
Cu. ft. air per ton iron and sulphur oxidized	96,145	142,224	156,000	98,847	139,361	112,950	118,328	119,475	240,937	123,583
Oxygen efficiency, taking into account total iron and sulphur oxidized	84.0	54.1	57.2	76.8	56.3	74.0	81.0	50.6	69.8	59.0
Total tons iron and sulphur oxidized per stand per month	2,075	1,610	.....	1,647	1,903	2,868	1,991	2,576	768	1,440
Punchers used per shell per shift	2	2	2	2	2	2	2	1	1.5	2
Tons bullion per puncher per month	221	204	522	119	478	392	255	137	129	240
Tons iron slagged per puncher per month	192	153	...	169	533	251	171	156	147	
Average time of blow, hours and minutes	4–00	4–11	3–09	6–44	9–20	8–00	8–30	7–12	4–51	10–00
Average time to blow one ton bullion, minutes	28	23	14	54	45	19	28	28	51	90
Average tons bullion per blow	8.5	10.8	13.3	8.22	12.7	25.3	18.0	15.6	5.8	6.7
Average weight of matte per charge, tons	21.9	26.0	25.9	33.9	40.5	60.0	50.0	51.2	19.7	26.6
Tons ore charged per ton of matte	0.23	0.20	0.155	0.201	0.17	0.25	0.30	0.21	0.286	0.283
Tons iron slagged per ton of available silica	....	2.44	....	2.45	3.85	1.75	1.80	2.36	2.03	2.60
Size of converters	12 ft.	12 ft.	12 ft	12 ft.	12 ft.	10 by 26 ft.	12 ft. Gt. Falls 10 by 26 ft. P.-S.	12 ft	7 ft by 10 ft. 6 in	9 ft. by 7 ft 6 in.
Number of tuyères	28	24	24	24	24	35	24 Gt Falls. 35 P.-S.	22	12	12
Size of tuyères, inches	1¼	1¼	1½	1½	1¼	1¼	1¼ Gt. Falls. 1½ P.-S.	1½	1½	1¼
Thickness of brick used on tuyère line, inches	24	30	30	14	30	18	18	24	18	15
Average slag analysis:										
SiO_2	19.6	21.7	17.2	17.4	15.6	23.6	25.7	29.2*	22.2	23.8
FeO	68.2	67.9	69.1	70.7	67.3	66.2	60.0	62.1	63.4	61.5
CaO	0.8	1.1	0.6	0.8	0.4	...	.....	1.6	0.9	
Al_2O_3	3.4	3.0	3.7	3.0	2.9	...	.....	....	2.8	
Average matte analysis.										
Cu	37.7	43.37	51.2	28.22	34.7	43.7	42.8	31.9	35.21	25.0
Fe	32.5	29.2	21.6	30.9	34.9	28.0	27.4	28.8	36.1	46.0
S	27.4	23.5	24.5	25.3	24.9	25.1	25.6	18.7?	25.0	25.0

1. Arizona Copper Co., Clifton, Ariz.
2. Old Dominion Copper Mining & Smelting Co., Globe, Ariz.
3. International Smelting Co., Inspiration, Ariz.
4. Calumet & Arizona Mining Co., Douglas, Ariz.
5. Copper Queen Mining Co., Douglas, Ariz.
6. Consolidated Kansas City Smelting & Refining Co., Hayden Plant, Ariz.
7. Consolidated Kansas City Smelting & Refining Co., El Paso Plant, Texas.
8. Cananea Consolidated Copper Co., Cananea, Sonora, Mex.
9. Detroit Copper Mining Co., Morenci, Ariz.
10. Consolidated Arizona Smelting Co., Humboldt, Ariz.

* Insoluble.

arrived at, it was impossible to obtain any figures covering the converting operations.

There is no question about the superiority of basic over acid linings or, where the output of copper is sufficiently large to insure efficient operating, of the standard type over the smaller basic-lined converter. The inability properly to control the temperature of the charge in the small vessel, and the resulting destruction of the brick lining, are the most serious difficulties with which the operator using a small basic-lined converter has to contend. Throughout the Southwest, the preference has been for the standard upright rather than for the horizontal converter. The greater flexibility of the former, and the greater ease with which repairs may be made, strongly recommend it.

The operating data in Table 1 will give an idea of the working conditions and the results obtained at some of the large smelting plants in the Southwest. It is to be regretted that replies to questions pertaining to power consumption were at such variance that it was thought best to omit them entirely, as no reconciliation seemed possible.

TABLE 2.—*Old Dominion Copper Mining & Smelting Co.*

Campaign of basic-lined converter No. 2.	
Put in operation	June 27, 1913
Removed for initial patching	Dec. 7, 1915
Total hours blowing	13,734
Total number blows made	3,288
Average time of blow	4 hr. 11 min.
Total number taps matte	9,316
Total tons matte charged	85,578
Average weight of matte per charge	26 tons
Average copper content of matte	43.37 per cent.
Total tons bullion produced	35,431
Average time to blow one ton copper	23 2 min.
Average tons copper per blow	10.80
Average blast pressure	13.2 lb.
Average air used to blow one ton copper	178,411 cu. ft.
Average air used per minute	7,691 cu. ft.
Average air used per ton iron slagged	242,502 cu. ft.
Total ore fed	17,097 tons
Tons ore fed per ton matte blown	0.200
Magnesite brick used for repairs	None

Average Slag Analysis

	Per Cent.
Cu	2.1
SiO_2	21.7
Fe	52.2
CaO	1.1
Al_2O_3	3.0
S	1.8

Average Matte Analysis

	Per Cent.
Cu	43.37
Fe	29.2
S	23.5

Separate operating data for the Peirce-Smith and Great Falls converters at El Paso could not be obtained, so the figures in column 7 are a combination of results obtained in these different types. The figures in columns 9 and 10 are for former acid-lined shells now operating with a lining of magnesite brick. All figures are for periods of 1 month or more. Those in column 5 are for the year 1915. The data in column 2 cover a period of 29 months' operation—the life of one original lining to the time of first patching. Although some of the figures in Table 2 have already been given, this more complete summary of the campaign may prove of interest.

I am greatly indebted to the managers and superintendents of the different properties who so graciously and promptly furnished me with the necessary information for this article, and take this opportunity to thank them.

Discussion

The Chairman (Walter Douglas, New York, N. Y.).—I presume, gentlemen, that Mr. Howard's experience with the Great Falls basic-lined converter has been as regards the tonnage produced from a single lining, perhaps, a world's record. About 6 months ago Mr. Howard shut down and tore out what was left of the first basic-lining which was installed at the Old Dominion, the lining having produced something over 70,000,000 lb. of copper. This was due probably not only to the very excellent grade of the matte which was produced by the smelter there, but also to the fact that after the tonnage began to reach record proportions Mr. Howard lived with that lining night and day, and watched it most attentively. May we hear from any of the gentlemen present with reference to this paper of Mr. Howard?

E. P. Mathewson, Anaconda, Mont.—I would like to say a few words in regard to the basic-lined converter. I noticed at El Paso that they had not adopted the Great Falls type of converter, but had stuck to the Peirce-Smith type of horizontal converter. This type has certain advantages over the Great Falls type and certain disadvantages. We have in the Northwest still larger furnaces of the Great Falls type, many of them 20 ft. in diameter. These furnaces are doing good work, due to the advantage that the Great Falls converter possesses. The sole disadvantage, I think, is that in tilting the furnace, if care is not taken by the operator, some of the tuyères are under a greater depth of matte than others; there is liable to be splashing and loss of blast from this same cause. In the horizontal type the tuyères are at the same level, and the level can be adjusted to meet the blast available, so that very little splashing results. There is a disadvantage on the other hand with the horizontal type converter in that it is hard to distribute the silica

evenly over the matte without making several openings. This difficulty is not met with in the Great Falls type, as the silica may be dumped without care, and will distribute itself evenly over the matte. But it will be very interesting, as I say, to note the comparison between these two types on a similar grade of matte. As I understand, the matte used at El Paso is about the same grade as used at the other points in the Southwest and about the same as in the Northwest. We have records of our furnaces extending back over 2 years, and I think we have forgotten that these linings have any life limit. A little care—which means that the operator must not be careless—keeping the heat down to reasonable limits, and you will not have difficulty with the lining. The main part of the lining we seem to be able to make last indefinitely, care being taken to keep the silica low in the slag.

Kuno Doerr, El Paso, Texas.—I am sorry to say that the new converter at our El Paso plant was put into use only about 10 days ago, due to the fact that we have had difficulty in getting magnesite brick to line it with. This converter is of the P-S type, 13 ft. in diameter by 30 ft. long, whereas our old Peirce-Smith converters are only 10 ft. in diameter by 26 ft. long. In the few days in which the new converter has been in operation we have observed a great improvement in it over the operation of the old converters. It has a large stack, or gas outlet, and makes practically no splash at all—that is, what splash there is goes back into the converter itself, whereas in our old converters we have had a great deal of trouble from accretions in the stack and hoods, and there has been a very large amount of cleanings to take up and resmelt. Thus far we have had none of this in operating the new converter.

Our average blister production per lining in the 10-ft. Peirce-Smith converters was about 7,000 tons. We have never had to reline the bottoms of any of our converters, and our repairs usually consist of relining with new brick in the tuyère zone only. A patch made without shutting down the converter except between blows is not considered in our records as a new lining, but any shutting down and cooling off of the converter for repairs terminates our production record per lining. Thus we actually get much more than 7,000 tons of blister on the average number of brick in a converter lining. Our mattes vary somewhat, but will not run much under or much over 40 per cent. copper.

One source of trouble with our old converter installation was the fact that the converters were not served by electric cranes, but all the matte from the furnaces to the converters was transported in open ladles over our industrial railway, and the run from furnaces out onto our slag dump and back again, to get the necessary height, was about ½ mile. The consequent cooling, and pouring of matte into converters through launders caused some of the troubles we had from accretions and excess-

ive amount of cleanings; but the greater diameter of the new converter, larger gas outlet, and better hood, are principally responsible for the improvement we have accomplished.

Mr. Howard's exceptional record of long life of linings, I imagine to be partly because he has had the pick of such converter silica as he has wanted to use, whereas we, being in the customs smelting business, have had to take what we could get.

You will thus observe that we have no production records as yet that would be of any interest concerning our new converters, but next year at this time we hope to be able to give a good account of our big converters. Our 10-ft. Peirce-Smith converters consumed about 9,000 ft. of free air per minute at, say, 12 lb. pressure, and the last records I secured before leaving El Paso show that our new 13-ft. converter is taking 20,000 ft., and that means that it is doing business.

Leaching Tests at New Cornelia

BY H. W. MORSE, PH. D., LOS ANGELES, CAL.,* AND H. A. TOBELMANN,† AJO, ARIZ.

(Arizona Meeting, September, 1916)

Introduction

The experimental work on the oxidized copper ore at the New Cornelia mine at Ajo, Ariz., ended on Jan. 12, 1916. On that date final decision was made on the general nature of the process to be used in the 5,000-ton leaching plant, and on many of the details, as far as experience on a 40-ton scale could decide them.

With the approval of the Board of Directors and the General Manager, John C. Greenway, we have compiled what seem to be the most interesting data on the results obtained during the experimental period. The most important part of the data resulted from the operation of a 1-ton and a 40-ton plant at Ajo.

Experimental work on Cornelia oxidized ore dates back to April, 1912, and has been going on nearly continuously since that time. A good many variations from the original idea have been tried, but the process finally decided upon is in principle and in all of its details a simple one.

The history of the leaching work on this ore has been brought down to about a year ago in papers presented to this Institute by Stuart Croasdale[1] and Dr. L. D. Ricketts.[2] The present paper will, therefore, deal principally with the results obtained during the last year of the work.

Preliminary Tests by Mr. Croasdale

Preliminary tests were begun by Stuart Croasdale in July, 1912. A number of important points were definitely decided by his work. Among these were:

1. New Cornelia oxidized ore, not very finely crushed (to about 2- to 3-mesh) showed good extraction with 5 per cent. sulphuric acid. It appeared that better than 80 per cent. extraction could be expected.

2. The consumption of acid for this extraction was not prohibitive. It was apparently 3.5 to 4 lb. gross per pound of copper recovered.

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[1] Leaching Experiments on the Ajo Ores, *Trans.*, vol. 49, p. 610.

[2] Some Problems in Copper Leaching, *Trans.*, vol. 52, p. 737.

3. It was proposed to make cement copper. The iron consumption per pound of copper recovered was a little over 1 lb.

4. A column 12 ft. deep of ore crushed to this fineness could be successfully percolated.

During Mr. Croasdale's work the plan in view was a very simple one indeed. The ore was to be leached with sulphuric acid. Cement copper was to be produced by the use of metallic iron. As Dr. Ricketts explained to the Institute nearly 2 years ago, the oxidized Cornelia ore could apparently be leached and cement copper produced at a profit, if sulphuric acid and scrap iron were bought in the open market and shipped to Ajo, and cement copper shipped out, all solutions being thrown away after removal of the copper.

Mr. Croasdale showed also that the amount of substances other than copper, taken into solution under his conditions of crushing and leaching, is small. This naturally suggested the possibility of a closed leaching cycle with electrolytic deposition and at least partial regeneration of the acid required for solution of the copper. Early experiments showed that iron, which is dissolved from the ore to a certain extent, and oxidized at the anode during electrolysis, was a troublesome factor.

There seemed to be three possible ways of meeting this difficulty: (1) The use of a diaphragm; (2) the use of a depolarizer; (3) purification from iron before electrolysis.

Antisell Process

About this time there came to the notice of the manager good results that were being obtained with a special anode, the invention of F. L. Antisell. This anode was in the form of a long, deep, narrow box with wood veneer sides, containing graphite electrodes and having the space between graphite and wood filled with selected coke.

A cyclic process was tried with these anodes. Ore crushed to 4-mesh was leached 3 days by upward percolation, with a practically constant solution containing 3 per cent. H_2SO_4, 2 per cent. ferrous iron, 0.5 per cent. ferric iron and 2 per cent. Al_2O_3. Sulphur dioxide gas was used to keep down the ferric iron concentration.

The process gave good results. The anode was, however, rather cumbersome and the absorption of the sulphur dioxide was not satisfactory, presumably owing to the rather high acid content of the solution.

The following is a summary of average results obtained by the Antisell Process: Tons of ore leached, 75; heads, 1.68 per cent. Cu; extraction, 77.6 per cent.; current density, 6.54 amp. per square foot; voltage, 1.23; production of copper, 1.72 lb. per kilowatt-hour; H_2SO_4 in solution, 2.73; sulphur per pound of copper, 1.06 lb.; total days run, 40.2; total cathodes produced, 803 lb.

Process of Pope and Hahn

While these tests were being carried on, another process was being developed at Raritan by F. A. Pope and A. W. Hahn. The Antisell idea was in a sense a combination of diaphragm and depolarizer, without any attempt to remove the iron. Pope and Hahn planned to remove the iron (and most of the other impurities as well) before sending solution to be electrolyzed.

Their process was cyclic. Ore crushed to 4-mesh was leached for 3 days with dilute sulphuric acid, the counter-current leach being so distributed as to permit of drawing off two classes of solution—one high in iron, the other low. The high iron solution was heated to about 200° F. with the addition of the requisite amount of finely ground (90 per cent. through 200-mesh) copper oxide in the form of roasted ore. The mixture of solution and oxides was agitated for 3 or 4 hr. and then put through a filter press. The result was the removal of nearly 90 per cent. of the total iron and 70 to 80 per cent. of the alumina from solution. An electrolyte high in copper and low in all impurities was produced and the precipitate, as filter cake, was granular and easy to wash and handle.

The other portion of the leaching solution, low in iron, was electrolyzed and returned to the cycle until its iron content was high enough to require purification.

Results of the Pope-Hahn process were very satisfactory, but the plant had hardly started when the war broke out and all experimental work was stopped. The following summary will indicate the results obtained: Ore leached, 329 tons, 10 charges; average copper heads, 1.12 per cent.; copper tails, 0.30 per cent.; water, tails, 9 per cent.; copper per kilowatt-hour, 1.0 lb. About 40 lb. of roasted Miami concentrate per ton of ore handled took care of impurities; 92 per cent. of the copper in the concentrate was recovered.

Working the Pope-Hahn process would have required the installation of roasting plant, fine-crushing plant, agitators and filter presses and the purchase of high-grade sulphide ore or concentrates. It offered decided advantages, among them a practically pure solution for electrolysis, together with the regeneration of a high percentage of the acid necessary for leaching.

Greenway Process

Before the study of the effect of impurities had progressed very far, an interesting fact was noticed. If neutral solution from the leaching system was circulated through fresh ore, a large part of the iron in solution was precipitated, presumably as a basic sulphate. This basic sulphate was apparently not readily soluble in a solution containing more

acid. A possible method of purification from iron seemed to be indicated and the results of small-scale tests were so encouraging that a 1-ton plant was built. In the first tests, ore crushed to about ½ in. was leached for 6 days. The oldest charge of ore was in contact with fresh leach from the electrolytic cells. This leach carried about 3 per cent. of free H_2SO_4. Solution passed in counter-current through six tanks containing ore, and the acid of the leach was neutralized by the time it reached the fourth or fifth tank. In the last two tanks a neutral solution was in contact with fresh ore and here the iron was deposited.

The results obtained will be evident from the data of Table 2. The ferric iron was under good control for more than 100 complete cycles.

TABLE 1.—*Analysis of Average Cornelia Oxidized Ore*

	Per Cent.
SiO_2	65.7
Al_2O_3	15.3
Fe total	4.5
CaO	0.6
MgO	1.6
MnO	0.14
Cu	1.50
	Oz per Ton
Au	0.005
Ag	0.20
	Per Cent.
NaO	3.6
K_2O	4 6
P_2O_5	0.3
Cl	trace
CO_2	1.3
	99.04

At this time it was already practically settled:

1. That a good extraction could be obtained with dilute sulphuric acid.

2. That the acid consumption was not prohibitive.

3. That electrolytic copper could be produced successfully.

There remained, however, some important problems to be solved. Among these were:

1. Crushing; from two points of view: (1) for percolation in a deep bed; (2) for extraction.

2. Further study of the fouling of the solution, including the general accumulation of soluble salts, but especially those which might affect extraction or power efficiency in electrolysis.

3. Circulation and general handling on a small commercial scale.

4. Any factors other than those under (2), which might affect electrolysis itself or the quality of copper produced.

TABLE 2.—*Summary of All 1-Ton Tests. Aug. 8, 1914, to Nov. 3, 1915*

Charge No (Inclusive) Date	1–19 8–8 to 8–31	20–74 9–1 to 10–19	75–100 10–31 to 11–25	101–116 11–26 to 12–10	117–217 12–11 to 3–27	218–259 4–20 to 5–31	260–275 6–1 to 6–16	276–299 6–17 to 7–11	300–330 7–12 to 8–11	341–413 8–18 to 11–3
Average:										
Heads—per cent copper	1.70	1.22	1.55	1.39	1.29	1.30	1.36	1.29	1.22	1.33
Tails—per cent copper	0.64	0.29	0.43	0.32	0.24	0.24	0.19	0.17	0.24	0.27
Extraction—per cent	62.8	75.9	72.2	77.1	81.4	81.5	85.8	86.8	80.3	79.7
Per cent ore on 4-mesh	29.0	47.0	43.4	28.0	33.0	25.0	26.0	25.1	35.1	39.0
Days leached	6.0	6.0	6.0	6.8	7.6	8.0	7.5	7.5	7.5	8.0
Circulation—1-1 gal. per ton per min	2.0	1.8	3.3	3.3	3.0	2.0	2.0	2.0	2.0	2.0
Advance—1-1 gal. per ton per min	0.42	0.40	0.17	0.20	0.20	0.16	0.16	0.20	0.16	0.29
Electrolytic data:										
Current density, amp. per sq. ft.	9.8	12.0	7.0	7.3	7.1	6.9	9.2	6.6	5.3	8.5
Volts—drop in tank	2.11	2.36	2.00	2.15	2.12	2.29	2.30	2.08	2.03	0.97
Lb. copper per kw.-hr	0.98	0.83	1.08	1.01	0.99	0.77	0.76	1.07	0.95	2.0
Ave. free H_2SO_4 in cells	4.2	3.0	2.8	2.7	2.9	2.9	3.3	3.4	2.8	3.8
Ave circulation in cells	3.8	4.6	4.4	4.2	5.8	7.0	8.2	7.5	7.8	5.0
Ampere efficiency	83.7	76.8	83.5	85.0	83.1	67.2	67.3	75.0	74.0	69.5
Acid consumption:										
Total lb. 100 per cent H_2SO_4 used	1,217.0	1,160.0	729.0	467.0	3,428.0	1,107.0	492.0	1,002.0	1,113.0	4,240.0
Acid per lb. cathode Cu	5.33	1.76	1.16	1.62	1.66	1.46	1.30	2.01	2.74	6.65
Acid per lb. Cu leached	3.02	0.94	1.06	1.36	1.61	1.24	1.31	1.86	1.83	2.30

NOTES.—Charges Nos. 1 to 19 had no crushing plant, so screened rejects from sampling oxide pits were used. Ore had excess fines and contained some sulphides. No filter bottoms in tanks. Segregation, started with new solution. Leaching started Aug. 8, electrolytic end started Aug. 17 with grid lead anodes

Charges Nos. 20 to 74, on Oct. 6, tried reduction with liquid SO_2 using filtros bottom to distribute gas. Test stopped Oct. 14.

Charge No. 64.—All other charges normal. On Oct. 19, decided to discard one-half of solution volume, dilute to normal volume with well No. 1 water and make special run

Charges Nos. 75 to 100.—No SO_2 used and water from well No. 1 (high Cl) used on these charges.

Charges Nos. 101 to 116.—Same solution, longer leach, finer crushing variable conditions—No SO_2.

Charges Nos. 117 to 217.—On Dec. 14, SO_2 reduction tests were begun under the direction of F. L. Antisell. Lead-coke anodes were used. Tests discontinued Dec 26—between charges 117 to 127. Leaching varied from 7 to 13 days. Charges Nos 127 to 217 were run under normal conditions

Charges Nos 162, 163, 164 were high-grade ore—"k-10"-av 2.7 per cent. Cu These have not been included in the above averages. On Feb. 27, the average current density of 7 amp. per square foot was reduced to 4 amp per square foot.

Charge No 218—April 19 was started with 40 tons high ferric iron solution, other conditions normal.

Charge No 275—June 17, arrangements made for another SO_2 run with lead anodes, same solution used. Test continued until charge 300

Charges 300 to 330 run without SO_2 and under normal conditions

Charge Nos 341 to 413—New solution, carbon anodes, SO_2 reduction, air agitation

Ore used on charges Nos 20 to 413 was from open cut and averaged 1.307 per cent Cu

The 1-ton plant was therefore continued in operation and a plant to handle 40 tons of ore per day was built.

It is fortunate that this larger unit was built, for a good many things that had seemed easy to do in the 1-ton plant refused to work at all in the 40-ton. Fouling by iron and aluminum salts kept on increasing, the ferric-iron content of the solution rose to a point where the ampere efficiency in electrolysis was poor, and various undesirable phenomena appeared.

So the 40-ton plant was kept in continuous operation for nearly a year and all the factors that had been studied in the 1-ton plant were taken up, one at a time, and worked over again until finally it seemed that they could all be brought under complete control.

Final Leaching Process

The final process decided upon was as follows: Ore crushed to about 4-mesh; leached 8 full days by counter-current; washed with three counter-current wash waters (or possibly four); solution, practically neutral during last 2 days of contact with ore, is sent through reduction towers, where it meets sulphur dioxide in counter-current. The ferric iron is thus reduced to below 0.4 per cent. Thence it goes through a revolving tumbler in contact with cement copper. There the ferric iron is still further reduced and a corresponding amount of cement copper passes into solution. From the cement-copper tumbler the solution passes to a settling pond and then into the electrolytic cells, where a part of the copper is removed; then back into the leaching system, passing first through the oldest ore which has already been leaching for 7 days, and so on to begin the cycle again.

Important Factors

In a cyclic process such as this, no single factor or step can be said to be the most important. We can, however, consider a list of those factors that were studied with especial attention and can offer definite figures on the results to be expected.

Extraction

Both in the 1-ton and the 40-ton plants, it was easy to obtain 80 per cent. extraction, using about 3 per cent. sulphuric acid and 8 days of counter-current leaching by upward percolation on 4-mesh material. An extraction of 83 per cent. was reached over considerable periods of time. The continuous attainment of this figure was limited by other factors than the nature of the ore or the size to which it was crushed, such

as the fouling of solution, with the result that salts carrying copper actually crystallized in the ore in the leaching tanks and were not completely dissolved by washing.

Incomplete washing was, of course, another reason for low extraction under otherwise normal conditions, and the possibility of improvement in this point is clearly shown by the wash-water data of Table 6.

Keeping well within the time limit for washing set by the fixed cycle, it should apparently be possible to get an extraction of 82 per cent. or better.

Acid Consumption

The 1-ton plant, over its whole life of 413 cycles and under all sorts of favorable and unfavorable conditions, showed a net consumption of 1.65 lb. of 100 per cent. H_2SO_4 per pound of copper leached. This was somewhat lower than the figures given by the preliminary tests on the ore.

The 40-ton plant apparently did not do so well. During its 301 cycles, the average net consumption was 2.8 lb. of acid per pound of copper leached. The difference in favor of the 1-ton plant is probably due to the large leakage and general losses in the 40-ton plant.

In designing the plant which is now being built at Douglas to furnish acid for Cornelia, 3 lb. has been taken as a conservative figure for acid consumption.

Power Consumption for Electrolysis

In the 1-ton plant, with lead anodes, the average was 0.934 lb. copper per kilowatt-hour at 8 amp. per square foot of cathode surface. This was for all the conditions, good and bad, under which the plant was operated. With graphite anodes, also under all sorts of conditions, the figure was 2 lb. copper per kilowatt-hour at 8.5 amp. per square foot.

In the 40-ton plant, with lead anodes, at an average current density of 6.9 amp. per square foot, the average production under all conditions was 0.87 lb. copper per kilowatt-hour. With graphite anodes, the figure was 1.65 lb. It should be said, however, that the ferric iron was never under proper control during these tests with graphite in the 40-ton plant. During series VI of the 40-ton plant, with current density 6.4 amp. per square foot, the copper yield was 1.04 lb. per kilowatt-hour. This run was with lead anodes and under fair conditions, so far as ferric iron was concerned. In the final run with lead anodes, and with both ferric iron and general fouling under still better control, an even higher figure was obtained.

Lead anodes will be used in the big plant. The reasons for this choice will be given later.

TABLE 3. *Summary of All 40-Ton Tests with Lead Anodes. Feb. 1 to Nov. 8, 1915*

Charge No. (Inclusive) Date	1 to 81 Feb. 1–May 21	91 to 130 June 1–July 12	131 to 154 July 13–Aug. 4	155 to 170 Aug. 5–Aug. 22	171 to 201 Aug. 23-Sept. 25	202 to 238 Sept. 26–Nov. 7	1 to 238 Feb. 1–Nov. 7
Average:							
Heads—per cent copper	1.248	1.314	1.225	1.274	1.286	1.361	1.281
Tails—per cent copper	0.294	0.226	0.242	0.265	0.346	0.223	0.275
Extraction—per cent	76.4	82.8	80.2	79.2	73.1	83.6	78.5
Per cent. ore on 4-mesh	25.0	25.3	35.0	32.0	40.7	37.9	36.0
Days leaching	9.3	8.0	8.0	8.0	8.0	8.0	8.3
Circulation—gal. per ton per min	2.00	2.00	2.00	2.00	2.00	2.00	2.00
Advance—gal. per ton per min	0.20	0.35	0.35	0.35	0.35	0.38	0.30
Electrolytic data:							
Current density—amp. per sq. ft	7.4	7.1	7.1	7.1	6.0	6.4	6.9
Volts—drop in tank	2.43	2.20	2.18	2.04	2.18	2.07	2.24
Lb. copper per kw.-hr	0.72	0.99	0.85	0.98	0.86	1.04	0.87
Ampere efficiency	66.8	83.3	70.8	76.5	71.7	82.3	74.0
Lb. cathode copper	49,775.0	30,652.0	17,431.0	10,521.0	18,273.0	23,717.0	150,349.0
Acid consumption:							
Total lb. used—100 per cent	188,484.0	98,442.0	47,484.0	37,197.0	85,173.0	102,402.0	559,182.0
Lb. H_2SO_4 per lb. cathode Cu	3.82	3.21	2.72	3.53	4.66	4.31	3.71
Lb. H_2SO_4 per lb. Cu. in solution	2.88	2.69	2.39	2.58	3.67	2.98	2.89
Sulphur:							
Total lb. used	None	None	4,693.0	5,434.0	18,767.0	26,697.0	55,861.0
Lb. S per lb. cathode Cu	None	None	0.27	0.52	1.02	1.12	0.80
Lb. S per lb. iron reduced	None	None	1.50	0.56	1.20	0.99	1.08
	Old solution—No SO_2 first run	New solution—No SO_2 new run	SO_2 small tower acid electrolyte	SO_2 small tower neut. electrolyte	SO_2 large tower solution saturated	SO_2 large tower solution diluted	Av. of all 40-ton tests with Pb anodes

NOTE.—Forty-ton plant started Feb. 1, 1915, with Pope-Hahn solution, tank house started Feb. 26. Control of ferric iron lost on Mar. 12

Charge No. 26, various attempts made to change iron ratio. Low tank-house efficiency caused copper content of leaching solution to increase to such an extent as to interfere with the extractions. Troubles with pumps, talcy ore, linings, etc. Solution of these first 81 charges discarded on May 21 and a new series of tests started, May 22, 1915. Charge No. 91, new solution was made up for this run. Electrolytic plant started June 1. Iron control lost on charge No. 117. June 27, reduction of the ferric iron decided upon and on July 13 the acid electrolyte was circulated through small reducing tower. As reduction did not appear to gain on the oxidation in the tank house it was decided to circulate neutral solution through the tower. Results showed reduction of ferric iron in neutral solution to be less difficult than the ferric iron in acid solution. Large tower built, and completed Aug. 23. Acid electrolyte reduction continued through large tower. Solution becomes saturated with salts and on sudden changes in temperature deposits crystals on ore. Extraction decreases and it was found necessary to dilute the solution by intermittently "bleeding" the "high iron" tank and making up the deficiency with wash water. The average extraction on the 15 charges before bleeding was 68.7 per cent; the average extraction on the first 15 charges after bleeding, 80.7 per cent. On Nov. 7 carbon anodes were put into the tanks.

Control of Ferric Iron

Quite early in the study of the conditions for good electrolysis it became evident that the ferric-iron content of solutions must be kept within a low limit. The 1-ton plant took care of itself over a long period of time, as far as the ferric-iron content and the general fouling were concerned, without the use of sulphur dioxide or any other outside reagent. Iron and alumina were apparently precipitated on the ore.

The 40-ton plant started off in the same way. But before long the ferric iron began to go up and the ampere efficiency to go down. This is evident from the table showing general results on the 40-ton plant. Series II began with fresh acid and lasted 40 days. During this period 0.99 lb. Cu per kilowatt-hour was the electrolytic yield. The current efficiency during this period of low ferric iron and clear solution was 83.3 per cent. During period III the same solution was continued in use, but the ferric iron was not under control, and kept increasing. The analysis of the solution on July 22, in the middle of this run, is shown in column III of Table 4. The ferric iron content had risen rapidly from 0.180 per cent. on June 20 to 1.222 per cent. on July 22. The result was that during period III, the current efficiency was only 70.8 per cent. and the yield was 0.85 lb. copper per kilowatt-hour. From this time until the

TABLE 4.—*Solution Analyses*

Second Period of 40-Ton Plant Operation Beginning with Charge 91 in 1915

	I 6-1, Per Cent.	II 6-20, Per Cent.	III 7-22, Per Cent.	IV 8-22, Per Cent.	V 9-23, Per Cent.	VI 11-5, Per Cent.
Cu	1.11	2 26	2.50	3.39	3.47	2 95
H_2SO_4 free	4.03	2.88	2.56	3.15	2 40	3.05
Fe^{++}	0.09	0 176	0 720	1 49	2 14	2.52
Fe^{+++}	0.164	0.180	1.220	1.07	0.80	0.239
Al_2O_3	0.145	1.250	2.730	3.69	4.40	2.840
CaO	0.092	0.090	0.080	0 050	0.120	0.018
MgO	0.132	0.122	0.120	0.150	0.049	0.054
MnO	0.023	0.030	0 030	0.050	0.046	0.029
Cl	0 08	0 08	.....	.....	0.130	0.090
SiO_2	0.075	0.084	0.110	0.040	0.960	0.096
Specific gravity	1.100	1.150	1.250	1.370	1.376	1.330

Column I gives solution analysis at beginning of a run, where fresh acid has been used, and before electrolysis has been started.

Column III is the analysis of solution when reduction of ferric iron was begun.

Column V shows solution at maximum concentration. Cold nights caused the separation of a large amount of $FeSO_4$, $CuSO_4$, $24H_2O$.

Column VI is an analysis of solution under normal conditions, after "bleeding" regularly for about 40 days.

end of the experimental work, sulphur dioxide gas was used regularly to control the ferric iron. With lead anodes a low content of ferric iron is not necessary. The solution may contain 0.3 or 0.4 per cent. without any marked effect on the current efficiency.

Sulphur Consumption

To maintain satisfactory control of the ferric iron was, at this point in the work, the greatest apparent difficulty. To follow the course of the tests it will be necessary to keep in mind both Tables 3 and 4. The first of these shows the conditions under which the 40-ton plant was operating; the latter, the analysis of the solution at various times.

During period III, the acid electrolyte was circulated through a tower about 2 ft. square, in contact with sulphur dioxide. The acid electrolyte was taken from the electrolytic system and returned to the same system. The weight of sulphur that could be burnt and brought in contact with solution as SO_2 was too small, and the ferric iron kept on increasing. Under these conditions, we were burning 1.5 lb. sulphur per pound of ferric iron reduced.

Series IV began with only a single change, but this proved to be a very important one. The *neutral* advance from the leaching system was circulated through the SO_2 tower before entering the electrolytic system. Under these conditions, only 0.56 lb. of sulphur was required per pound of ferric iron reduced.

Although the small tower was not of sufficient capacity to make control easy, by Aug. 22 the ferric iron was on the down grade—1.07 per cent. as against 1.22 on July 22.

From this time on, as long as lead anodes were in use, the ferric iron did not get out of control, and on Nov. 5 the solution showed only 0.24 per cent. of this troublesome substance. The results with graphite anodes will be considered later.

Fouling of Solution

In the 40-ton plant the entire solution began to foul badly after about 100 charges had been leached. The cooler weather was beginning at this time and a bulky double sulphate of iron and copper crystallized out through the ore in the leaching tanks and plugged all the pipes and launders. To remedy this, a portion of nearly neutral solution was drawn off continuously from the leaching system and passed over scrap iron, making cement-copper. It was found that treating 1 per cent. of the total bulk of solution each day would maintain the fouling at a constant low point where it could cause no trouble. Columns V and VI of Table 4 show the effect of this "bleeding" of solution.

The cement-copper so produced was returned to a wooden tumbler and electrolyte on the way to the cells passed through it. Ferric iron is reduced to ferrous in this way and copper passes back into solution.

Under these conditions only electrolytic copper is produced and at the same time the control of the ferric iron is aided considerably. In the 40-ton plant it was apparently necessary to send about 12 per cent. of the total copper through this side cycle and back into the main one, to control completely the fouling materials introduced from the ore during leaching.

Grade of Copper Produced

Cornelia copper produced direct from the ore is the equal of any electrolytic copper produced in this country. Fortunately the orebody contains only minute amounts of either arsenic, antimony or bismuth. The analysis and the physical tests on Cornelia cathodes will be found in Table 5.

Table 5.—Analysis of Cornelia Copper Cathode

	Top, Per Cent.	Bottom, Per Cent		Top, Per Cent.	Bottom, Per Cent.
Cu..	99.900	99.868	S..	0.0369	0.0655
Fe..	0.0186	0.0291	As.	0.0013	0.0017
Cl..	0.0012	0.0019	Sb..	0.00095	0.0014

Physical Tests on Wire Bars

Tensile test	64,500
Elongation	1.9
Bends	36
Twists	43
Conductivity	100.7

Size of Material. Percolation

The work at Ajo was all done on the product from a Symons fine crusher, without any screening (except for tests on sized material). It was nominally a 4-mesh product but carried an average of about 30 per cent. of oversize. Extraction on this is satisfactory and while it could be increased by finer crushing, a large percentage of the copper minerals lies in fracture planes, and no very great profit would result.

This product is especially well suited to upward percolation in a bed 12 ft. deep or more. It does not segregate nor channel badly, drains well and is easy to wash.

Circulation in the Leaching System

In the 40-ton plant, each of the eight active tanks had a closed circulation, upward through the ore, of about 80 gal. per minute. This rate

is probably much higher than necessary, but for the experimental work it was made ample and retained as a fixed factor. Beside this closed circuit, which runs continuously, a portion of the solution from each tank is continuously advanced to the next tank in the series.

Fresh electrolyte from the cells, low in copper and of maximum acid content, leaves the electrolytic system and passes into the leaching tank containing the oldest ore—that which has been leaching for 7 days. Practically neutral solution, high in copper, leaves the leaching system at the tank containing new ore. Between these two points, and rigidly linked to them, is the advance from tank to tank.

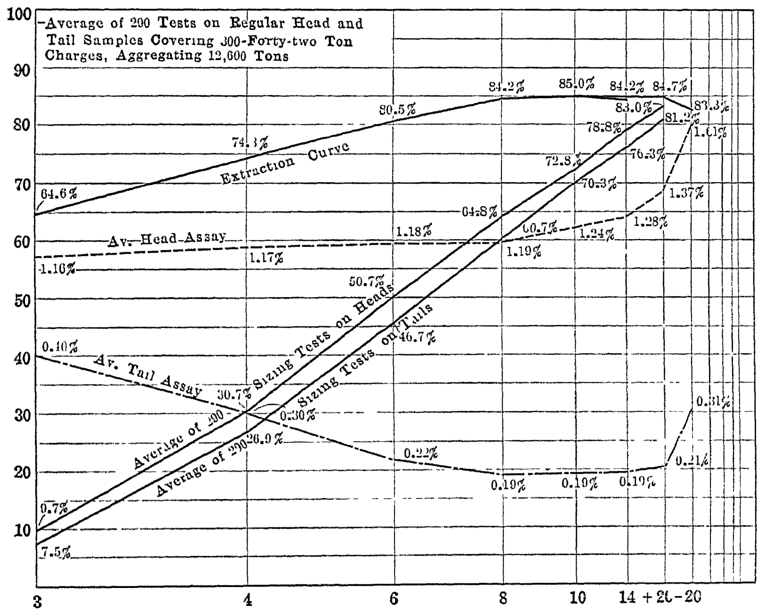

Fig. 1.—Sizing Tests on New Cornelia Oxidized Ore showing also Average Assays on Heads and Tails and Average Extractions on Sized Material.

It is of great importance to have a practically neutral solution leaving the last leaching tank and starting back toward the electrolytic system.

At the same time, we want as high an extraction as possible, which means as many tanks as possible with a fairly high acid content. To attain both these objects at the same time the advance from tank to tank is carefully regulated so that about six tanks of the eight active ones are acid practically all the time. The two others are neutral and the point of neutrality of the system just balances, varying possibly one tank in either direction during 24 hr., but returning to its original position.

Circulation in the Electrolytic System

The electrolytic cells, together with tanks and sumps for storage, form a second closed circuit from which solution is bypassed continuously into the leaching system at the same rate as the advance from cell to cell. To produce good, coherent cathodes, the circulation through the cells must be much greater in volume than the advance from the leaching system, as the latter is rigidly fixed by the necessity for maintaining neutral leaching tanks. While the advance in the 40-ton plant was about 12 gal.

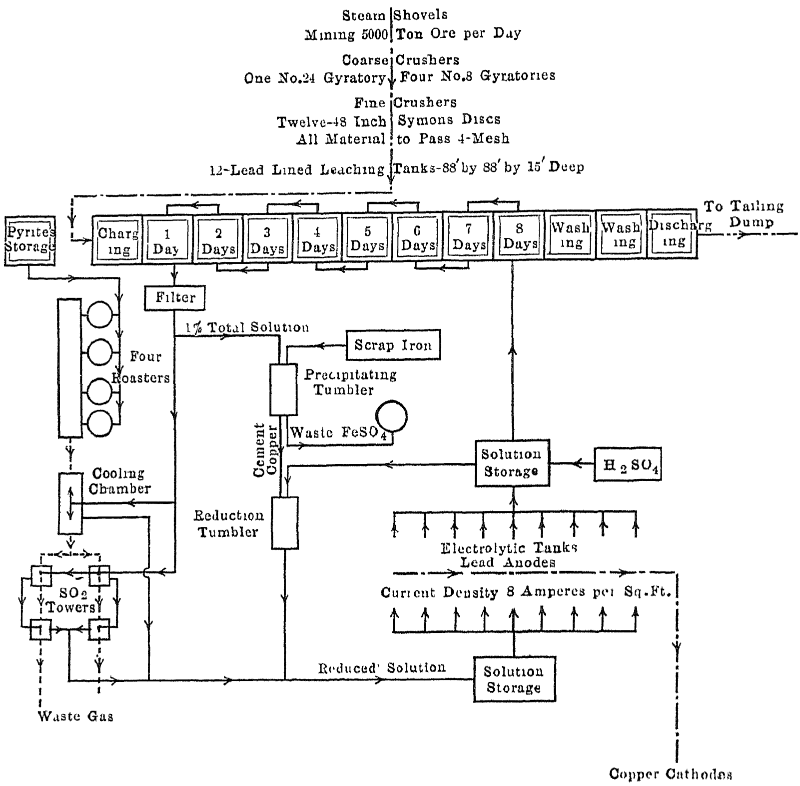

FIG. 2—FLOW SHEET OF 5,000-TON PLANT OF NEW CORNELIA COPPER CO.

per minute, it was advantageous to circulate in the tank house at rates up to 150 gal. per minute.

This circulation gives satisfactory results with lead anodes, but when graphite is used, rapid circulation alone is not enough to give good depolarization at the anode. Violent agitation in the electrolytic tanks is necessary to attain the lowest voltage when graphite anodes are in use, and this was accomplished by blowing air through small holes in lead pipes laid along the bottom of the cell.

Lead vs. Graphite Anodes

In making final decision on the process to be used, there were many mechanical questions to be settled. Crushing, method of percolation, circulation, pumps, structural materials, the handling of ore and tailings and many other points came up for decision. Besides these, there were some questions which bear more directly on fundamental principles. One of these was whether we should use lead or graphite as material for anodes.

Lead was in service for over a year and its behavior was well understood. But graphite promised much better results as far as power efficiency is concerned, and also as a means of regenerating a much larger part of the necessary acid for leaching.

The 1-ton plant gave most encouraging results with graphite over a period of continuous operation of nearly 3 months. With a current density of 8.5 amp. per square foot, over 2 lb. of copper per kilowatt-hour was produced, and during part of this time conditions were not good. The ferric iron was not under complete control. When all conditions were right, we made 2.25 to 2.40 lb. copper per kilowatt-hour; on some days as high as 2.75 lb. This was during the warm weather at Ajo, when electrolyte temperatures held well above 105° F. The 40-ton plant was using the large tower for reduction and the 1-ton plant used the small one. The SO_2 used in controlling ferric iron was bypassed from the main supply.

The 1-ton electrolytic plant was out of doors and while the odor of SO_2 was strong in the immediate neighborhood of the cell, it was not bad enough to prevent working over it.

By the time the graphite anodes were installed in the 40-ton plant, the average electrolyte temperature was only about 75° F. We could now determine accurately the efficiency of absorption and reduction by SO_2 (which we could not do in the 1-ton plant) and we could also examine carefully the circulation conditions necessary to control ferric iron.

The problem now began to look harder. As measured by the production of ferric iron from ferrous in the cell, the anodic efficiency of graphite, with rapid air agitation, is well over 100 per cent. With lead it is 30 to 35 per cent. We had, therefore, to combat about three times as much ferric iron per pound of copper with graphite as with lead.

Under the existing temperature conditions and with the rapid circulation that was necessary in the electrolytic circuit, the reduction efficiency of the SO_2 was only about 20 per cent. We could not control the ferric iron without either allowing the electrolyte to stand for a good many hours between the SO_2 towers and the cells, or else heating it to about 150° F. during its passage from towers to cells.

The absorption of SO_2 in the towers was perfectly satisfactory. But

unless it was used to better advantage in the reduction of the iron in the solution, it was impossible to live in the tank house without either very careful ventilation or a diver's outfit. With the violent air agitation that was necessary for high power efficiency, most of the SO_2 which had been absorbed in the towers was blown out in the tank house.

These difficulties are by no means insuperable. Of more fundamental nature is the low current efficiency obtained with graphite and its extreme sensitiveness toward ferric iron. When ferric iron is practically completely removed by careful reduction with SO_2, graphite can be made to give high current efficiencies. But the very slightest rise in ferric-iron concentration results in an immediate and considerable lowering in ampere efficiency. Even in our good runs with graphite in the 1-ton plant, the current efficiency was not much above 70 per cent. With lead anodes, and without any particular attention to the ferric iron, which was usually 0.3 or 0.4 per cent., we often obtained current efficiencies of 90 per cent. or better over considerable periods.

It may be of interest to state some of the points that enter into a decision between the two kinds of anodes:

1. The first cost of an installation is about the same.
2. The life of lead is known with some accuracy. The life of graphite in this service is not known.
3. The salvage value of lead is high. With graphite it is practically nothing.
4. The probable power saving to be expected (graphite over lead) if graphite were successfully operated, is about 0.6 kw.-hr. per pound of copper.
5. More than twice as much acid is regenerated with graphite as with lead.
6. Circulation must be more rapid with graphite.
7. Agitation during electrolysis is absolutely necessary with graphite.
8. The solution must stand for some time (probably 16 to 24 hr.) or be heated to 150° F. or higher, to obtain even reasonable SO_2 efficiency with graphite. This is not necessary with lead (see note under 10).
9. Much larger towers are needed when graphite is used.
10. The volume to be pumped to the top of the towers is larger with graphite.

(These last two statements contain the fact that acid electrolyte must be sent through the towers. The iron content of the solution in the cells must not rise above 0.2 per cent., if good efficiency is to be reached with graphite. With lead anodes it may go to 0.4 or 0.5 per cent. without much of any effect on power consumption. With lead it is sufficient to send the neutral advance to the towers. With graphite not only this volume, but also a much larger volume from the tank-house circulation which must be kept low in ferric iron must be sent.)

11. The tank house for graphite must be carefully ventilated.

12. The tank house for graphite must be larger (about as 9 to 7) on account of the lower current efficiency.

13. Operation with graphite requires the closest attention to every detail of reduction, circulation and agitation. Operation with lead is practically fool-proof.

This last point is probably the most important of all. There are enough so-called "minor" troubles about a leaching plant.

Control of Ferric Iron in Acid and Neutral Solution

During the periods when acid electrolyte was being sent to the tower for the reduction of ferric iron by sulphur dioxide, the sulphur efficiency was low. It proved to be about 20 to 25 per cent. Two causes at least combine to produce this result: First, the low solubility of SO_2 in acid solution, and second, the slowness of the reduction reaction in acid solution. Neutral solution was sent to the towers in later runs and on this the sulphur efficiency reached 75 to 80 per cent. That is to say, of the actual weight of sulphur burned to SO_2, 75 per cent. is utilized in the reduction of ferric iron to ferrous.

Washing the Ore

Each charge was given three washes. The first was the second wash of the preceding charge; the second, the third wash of the preceding charge; the third, fresh water. The wash which had been used three times was run into the electrolytic system. One complete fresh wash water was just sufficient to balance the loss from the system in the tailings and by evaporation.

The summarized wash-water data are given in Table 6 for the solution which was started in the 40-ton plant May 22, 1915. The most significant thing in this table is the effect of fouling as shown in the second column. While the solution was fresh, washing in the prescribed manner, the extraction was 81.1 per cent. and soluble copper left in the tailings was 0.045 per cent. When the solution became foul, even though a little more fresh water was used, extraction dropped to 73.5 per cent. and the soluble copper in the tailings was 0.103 per cent. Later, when fouling was being reduced by "bleeding" and additional wash water could be used in the same cycle, the extraction rose to 83 per cent. and the soluble copper in the tails dropped to 0.033 per cent.

It is of course possible to wash tailings with extra water, passing this over scrap iron and making cement-copper, and this should naturally be done until the expense of operation balances the value of the recovered copper. The experiments at Cornelia were carried no further than is shown in this table.

All the experimental work in the 40-ton plant was carried out during continuous operation, so that the result should correspond as closely as

possible with commercial conditions. The regular tonnage was crushed each day and charged into the leaching tank. Tailings were removed, and circulation, advance, washing, electrolysis and reduction carried on without a break. Any change in operation was carried on for at least a week, and usually for 10 days. If the change promised to give improved results, it became a fixed part of the regular operating routine and the next point came up for study.

TABLE 6.—*Wash-Water Data*

Charges Covered by Period	No 91 to No 170, May 22 to Aug 20	No 171 to No 205, Aug 21 to Sept 25	No. 206 to No 301, Sept 26 to Jan. 10
Tons of ore treated	3,376	1,435	4,051
Per cent. copper in the heads	1 278	1 282	1 331
Per cent copper in the tailings	0 240	0 342	0 237
Per cent extraction	81 1	73 5	83 0
Per cent. copper in washed tailings	0 195	0 239	0 204
Per cent. extraction if tailings had been washed clean	84 7	81.3	85.3
Average moisture in tailings	10 52	9 57	9.65
Average number of washes	3	3	3
Total gallons new water used per ton of ore leached	40 6	42 2	59.1
Gallons entrained in tailings per ton tailings discharged	24 2	21 2	21.8
Gallons unaccounted for per ton of ore leached, *i.e*, evaporation and leakage, etc.	16 4	21 0	12 1
Gallons of solution bled per ton ore	None	None	25 2
Specific gravity of 8th day solution	1.100 – 1.290	1 290 – 1 400	1.400 – 1 300
Specific gravity of last wash	1.050 – 1 060	1.060 1 150	1.150 – 1.050
Average per cent. copper in first wash	1 66	2.96	2 25
Average per cent. copper in second wash	1 17	2.17	1.45
Average per cent. copper in third wash	0 61	1.25	0.67
Average per cent. H_2SO_4 in third wash	0 35	0 35	0.35
Pounds soluble copper per ton tailings	0.9	2 1	0.7
Probable extraction if one more wash had been used and half of this soluble copper had been recovered. Per cent.	83 3	79.0	84.2

Average of whole run:

Per cent. copper in heads	1.338
Per cent. copper in tailings	0.256
Per cent. extraction	80.7
Per cent. extraction—according to washed tails	84.1
Per cent. extraction—if 4th wash had been used	83.0
Per cent. moisture in tailings	10.0
Average gallons of solution entrained per ton of tails	22.6

Using this method, when we were through, we had a definite set of conditions under which known results could be produced for any length of time. There were no odds and ends to go back and pick up. No "minor" problems, which might turn out on investigation to be fundamental, remained to be solved. Mr. Greenway saw that this plan was strictly carried out.

During the experimental work, something over 180,000 lb. of perfect cathodes were produced, besides a good deal of cement-copper.

Summary

So far as the Ajo tests can show what will happen in a 5,000-ton plant, the following may be expected:

Extraction on 1.4 per cent. ore using closed wash cycle, 82 per cent.

Extraction on 1.4 per cent. ore using extra wash water, 83 per cent.

(The extraction figure used in making estimates has always been 80 per cent.).

Acid consumption (100 per cent. H_2SO_4), 3 lb. per pound of copper. "Bleeding" to prevent fouling of solution.

Power efficiency in electrolysis with lead anodes, 1 kw.-hr. per pound of cathode copper.

Sulphur consumption, 0.5 lb. per pound of cathode copper.

Neutral electrolyte only will need to pass through the reduction towers.

About 1 per cent. per day of the total solution volume must be sent over iron to control fouling.

About 12 per cent. of the total copper produced will pass through the cement-copper side cycle.

The circulation used in the test-plant leaching tanks (2 gal. per minute per ton of ore) is undoubtedly much higher than necessary. It is probable that a small part of this volume would give equally good extraction results

Discussion

The Chairman (H. W. Morse).—Gentlemen, for the first time in the history of the American Institute of Mining Engineers, we have a full session on the subject of leaching—especially on the leaching of copper ores. This branch of metallurgy is rather in its infancy. We have had one big plant running for a year or more, and pretty successfully; and another one which will be started in February or March, the New Cornelia. We have another one at Garfield for the Utah company, which is just designed and construction begun; and from several other points come reports of interesting test work. The first paper on the program this evening is one of some of the test work on the New Cornelia ore by myself and Mr. H. A. Tobelmann.

LAWRENCE ADDICKS, New York, N. Y. (written discussion).—The Ajo proposition has had, as I understand it, two great advantages: it has not been obliged to measure leaching against flotation or some other metallurgical process, and it deals with a copper content high enough to pay for straight sulphuric acid-iron cementation work if forced back upon such a plan.

These advantages have made it possible to prosecute a consistent policy of development along straight hydrometallurgical lines.

The history of these experiments shows a series of investigations into more or less complicated schemes with a gradual return toward simplicity, the present plan involving utilization of SO_2 in regular acid-making practice, precipitation of excess impurities upon the ore itself, final control by cementation and recovery of the copper by the use of lead anodes along more or less copper-refinery lines.

This is about as simple a scheme as could be devised in true cyclical form, and while I think no one will dispute the wisdom of adopting just such a scheme as the first step in large-scale operation, I venture to predict that as experience is gained in actual practice the tendency will be to return to greater complication, as one item after another can be examined separately and tested for cost. The great difficulty in launching a leaching process lies in the multiplicity of interrelated problems which must at first be dealt with jointly. Also in the fact that no two ores seem to present identical problems for solution.

I think this feeling I have outlined is exemplified by the discussion of the reasons for choosing lead anodes as given in the paper and I want to add a few words to this discussion.

The crux of the whole electrolytic problem lies in the mechanism of the oxidation and reduction of iron sulphate. A carbon anode must be thoroughly depolarized or the liberated oxygen will attack the anode itself and the graphite will disintegrate leaving a valueless sludge. In the case of a lead anode, lead peroxide and lead sulphate are formed but the disintegration is very slow as these substances form more or less adherent and conducting coatings. This means that with carbon anodes we impose at once the necessity of dealing with the entire equivalent of ferric sulphate if ferrous sulphate be the depolarizer. With lead anodes the issue can be largely but not wholly dodged through the escape of elemental oxygen. This in turn leads to the necessity of thorough circulation in the case of carbon anodes as against indifferent circulation in the case of lead.

Thorough circulation, however, results in intimate contact between the cathode and the ferric sulphate in the electrolyte with the natural and undesired reaction between these two substances.

Therefore, the price of our increased recovery of copper per kilowatt-hour is the obligation to keep liquors running more than some 0.2

per cent., and preferably much less, in ferric sulphate, away from intimate contact with the cathode, which means either a diaphragm or an efficient utilization of SO_2 or other reducing agent.

Everyone who has advocated the first remedy has come to grief, except possibly Hybinette in his two-level system in copper nickel refining, and everyone who has wrestled with the second problem has found that there are intermediate and as yet little-understood reactions to be dealt with. Nevertheless, I believe that as continued operation gives experience and opportunity for experiment, a way out of these difficulties will be found and the carbon anode yet succeed in the practical electrolysis of sulphate solutions.

G. D. Van Arsdale, New York, N. Y.—Both those responsible for the tests of which the results are given in this paper and the Institute are to be congratulated; the first for the successful results finally obtained on a difficult and new problem, and the profession on the liberal policy which made possible the publication of this very valuable summary of practical leaching data.

The thoroughness and care with which this work has been done may be judged from the time, nearly 4 years, during which these experiments have been carried on.

It is especially gratifying to the writer that some of the main points developed during the series of experiments on leaching by Phelps, Dodge & Co. at Douglas have been entirely borne out by this work; the first of these being that ferric iron can be controlled by the use of sulphur dioxide, the second that solutions that had previously been considered entirely too "foul" as an electrolyte for copper deposition can safely be used, and the third that by the use of depolarization the anode problem is entirely solved, either by lead or graphite, both of which had previously been considered as impossible for electrolysis of sulphate solutions of copper.

The paper is particularly valuable in giving a discussion of the conclusions leading to the adoption of the final process by the New Cornelia company and as bearing on their choice of lead instead of graphite as an anode material. My reason for discussing, and to some extent differing from, these conclusions is that I feel that from the results of Phelps, Dodge & Co.'s work it is necessary to emphasize the fact that, while the choice of lead in this case may be undoubtedly justified, yet there are conditions under which graphite should be selected and these conclusions therefore should not be taken as general conclusions against its use under other sets of conditions.

Graphite, which was first used under the writer's direction on a considerable experimental scale as an anode material for this kind of work, before this time had been considered as absolutely unsuitable for the electrolysis of sulphate solutions of copper. I have a letter from the manufacturers in which this statement is made, and the same thing is

tated in Greenawalt's book on copper leaching. There is no doubt that ts use has certain practical disadvantages, the principal one of these)eing that a tank-room ventilation system will probably be needed to :nable it to be used. While this means a radically different tank-room lesign, it need not mean anything at all impracticable or leading to ncreased tank-room costs beyond the negligible amount of power for ventilation. On the other hand, its use has very considerable points in ts favor which in my opinion outweigh those against it, generally speaking.

Regarding its durability, I believe it is entirely safe to say, from the esults of careful tests made by Phelps, Dodge & Co., that under proper and easily controlled conditions its depreciation costs, or in other words the amount of disintegration, per pound of copper produced will be negligible.

The ampere efficiency of graphite is, as stated, badly reduced by a erric iron content of electrolyte, with which still fair efficiencies may be had with lead, this meaning as stated the maintaining of a limit of about 0.2 per cent. ferric iron for graphite as against a limit of about).4 per cent. with lead. It should be carefully noted, however, that as egarding the amount of power per pound of copper produced a reduced ampere efficiency at a voltage of about 1 volt does not mean nearly as nuch as the same reduction with a voltage of over 2 volts. In other words, one of the very considerable practical advantages of graphite s the much lower voltage at equal current densities with lead. Regard-ng the ampere efficiency obtainable with graphite, it is possible that a asual reading of the paper under discussion might lead to the conclusion hat a low ampere efficiency was characteristic of graphite. That this s not true was proved conclusively by Phelps, Dodge & Co. in a series f tests last summer which demonstrated that it is commercially practical) keep the ferric iron under 0.2 per cent. and under these conditions to btain fully as good or better efficiencies than those cited in the present aper, with yields over a considerable period of 3 lb. of copper per kilo-att-hour at a current density of about 12 amp. per square foot, which should be noted is 76 per cent. in excess of the average current density s given in the table on page 837 of the paper under discussion. With ie lower current densities as given in this paper, our pounds of copper er kilowatt-hour would have been still higher.

It is, of course, not reasonable to expect that a high SO_2 reduction ficiency will be obtained if the SO_2 is blown out of the solution before has had time to act. In the worst case, this factor of the time needed r the reduction of ferric iron by SO_2 means a large, though not at all ohibitive, storage capacity, and it may be stated that one of the re-lts of our work at Douglas last summer was to demonstrate that the

ferric iron can be kept within our 0.2 per cent. limit by a storage of a portion only of the main solution.

It is true that the anodic efficiency of graphite is as stated, on account of air agitation, over 100 per cent., that of lead being 30 to 35 per cent. Expressed in another way, however, this means, with anything like a reasonable SO_2 reduction efficiency, which can be had as we have shown, that for practical purposes the amount of acid regenerated by lead is much less than with graphite. Now if one has an acid plant already built or ordered or intends to erect one, the comparatively small amount of acid produced by the use of lead anodes is not so important, but where this is not the case this considerable extra amount of acid produced by the use of graphite is a considerable practical advantage in its favor. Where, as in this case, an ore does not require more than 3 lb. of acid for leaching per pound of copper produced, the amount of acid regenerated by the use of the graphite system will be ample for leaching purposes. A comparison, therefore, between the two systems should include the cost of an acid plant against lead as regards installation costs, and, as regards operating costs, the extra cost of acid needed together with freight if brought from a distance. It seems to me, therefore, that the statement in point 1 of the summary on page 844 should include this factor.

Without intending criticism of the decision made in this particular case, I believe, nevertheless, that the following comparison as to the relative merits of lead and graphite will be true and more generally applicable than those given.

1. Installation costs for the two systems will be about the same, except that, where an ore requires more than 1½ lb. of acid per pound of copper produced, the additional costs of an acid plant plus railroad equipment for carrying it when produced at a distance from the leaching plant, plus storage tanks, piping, etc., at the leaching plant, must be charged against lead.

2. Careful determinations extending over a considerable period have shown that the disintegration of graphite and its consequent depreciation per pound of copper produced is negligible. The prevention of graphite anode disintegration means simply maintaining the conditions necessary for proper depolarization. When this is done graphite will not disintegrate appreciably, but if these conditions are not maintained it will go rapidly. Exactly the same thing is true for lead, but, since the anodic efficiency of lead is only one-third that of graphite, the disintegration of lead anodes under the same service may be expected to be at least as rapid as graphite.

3. Peroxidized lead detached from an anode will be lost, and the scrap value of partly peroxidized and badly corroded lead anodes, when high freight rates are considered, will not be very high. Furthermore,

when the slow rate of disintegration of graphite anodes finally reduces their thickness very appreciably, they can be reassembled and the scrap loss reduced thereby to a small amount.

4. The power required for precipitation with lead is at least three times that for graphite under the same conditions.

5. More than twice as much acid is regenerated with graphite as with lead. This, as stated above, means not only an extra installation but also an extra operating cost for this additional acid needed, when lead is used.

6 and 7. Since the anodic efficiency of graphite is very high, higher circulation and some form of agitation in the cells are required. In this connection it may be stated that, although air agitation has been used in the work at Douglas with good results in some respects, I have never been in favor of this method of agitation as compared with others, unless more than 3 lb. of acid per pound of copper are needed. The excess anode efficiency over 100 per cent. is of course mainly due to the oxidation caused by the air, and it is clearly illogical to use an oxidizing agent for mechanical purposes where reducing conditions are wanted.

8. The extra storage capacity needed for the time of reduction by SO_2 as shown above need not be more than a fraction of the total solution bulk. Furthermore, if a good SO_2 reduction is obtained, the tank-room ventilation difficulties are correspondingly reduced.

9. and 10. Since the anodic efficiency of graphite is three times that of lead, it of course necessarily follows that larger tower capacity and greater volume of solution sent to these are required. However, the saving in installation by not needing an acid plant will pay for this and the extra storage capacity needed under the preceding paragraph at least several times over.

11. The point of the necessary tank-room ventilation has already been spoken of.

12. I must disagree absolutely, so far as my experience goes, with this conclusion as to tank-house capacity being necessarily larger with graphite than with lead. While the cost of power is an important factor in this connection, we have not considered for our requirements that the low-current densities, averaging 6.8 amp. given in the paper, are economical under our conditions, and the figures we have obtained with current densities around 12 amp. per square foot, in which average ampere efficiencies at this higher density are fully equal to or better than those given in the paper and obtained over considerable periods appear to throw this comparison decidedly in favor of graphite.

13. It is a little difficult to see just why operation with graphite anodes will be any less "fool-proof" than with lead. Granted that the conditions for keeping the ferric iron below our limit are known, I cannot see any reason whatever for requiring any more care or skill for main-

taining this limit than for maintaining an equally vital one differing from this by only 0.2 per cent.

My opinion, not expressed as a criticism of the decision made in this particular case, but as a general one, is that the advantages and undoubted lower operating costs of graphite decidedly outweigh those with lead. In other words, it would only be considered advisable under very exceptional conditions to make the very large additional investment required for an acid plant over and above the acknowledged nearly equal costs of the two systems, and still with this extra investment to obtain a higher operating cost.

The Chairman.—I would like to say that Mr. Van Arsdale should be thanked for a good many discussions and arguments, and that the conversations and arguments with him were of very great advantage to us in this work at the New Cornelia; and in a good deal of what he says I would agree. It is highly probable that we will be able some day to work out the use of graphite to make a saving in power cost, and to generally improve the process. As far as we could see from the test at the New Cornelia, this was going to be a difficult undertaking, so we decided to put it off for a few years. I think we will later come back to large-scale tests on the use of graphite. Certainly the gain to be expected when this has been worked out is a great one. I, personally, was sorry to stop the research work on the graphite.

S. J. Jennings, New York, N. Y.—I would like to ask for some information. I see that this paper has considered merely two kinds of insoluble anodes, lead and graphite. I understand a considerable amount of work has been done with other insoluble anodes and I would like to ask if tests have been made at New Cornelia with any other insoluble anode such as fused magnetite.

The Chairman.—We have made none at Cornelia, except with lead and graphite, and a mixture of the two—that is, coke cast in lead.

F. S. Schimerka, Clifton, Ariz.—In connection with the paper presented, I wish to ask the question why the process of precipitating the ferric iron in a neutral liquor on fresh ore was not finally employed and preference given to the introduction of sulphur dioxide to effect the elimination of trivalent iron?

In regard to the use of lead anode sheets, I would like to know whether lead-covered iron grates would be applicable, and I ask this question because I remember that I have seen such anodes employed in liberator tank practice but have not been able to collect data concerning their efficiency compared with that of solid sheets used for the same purpose.

The Chairman.—That was the basis for starting the large plant. In the 7-ton plant and in the 40-ton plant, at first, the ferric iron was

precipitated, but later for an unknown reason we got no such result, the ferric iron going up and the ampere efficiency going down.

C. G. Grabill, Matehuala, Mex.—I would like to ask if it was possible that the difficulties arose from the depth of ore in the two tanks. In the first one you have a small volume of solution; and in the second, you have a very much larger body of solution. In the first condition, with the small volume, you have much more air present and oxidation taking place. If the charge is increased—rather, if the percentage of the solution is increased—that condition does not obtain in the latter part of the action. I would like to ask if you have done any work along that line—to investigate that point?

The Chairman.—That is one question we left to find out in the 5,000-ton plant. Mr. Flynn, have you anything to say?

F. N. Flynn, Clifton, Ariz.—The leaching that we did at Clifton, for something over 20 years, was the simple process of an acid leach, with iron precipitation, and the wasting of all liquors. The modern leaching which you are about to attempt is something which you have developed and proven to your entire satisfaction and until such time as it is working out, practically, I don't feel that I would have any criticism to make. I don't feel competent to criticize. It does seem to me that you are working in the right direction. The only question that comes to my mind is the percentage of liquors you will have to throw away. That, I understand, you have worked out to your entire satisfaction.

The Chairman.—I think that 6 months' operation of the big plant will make us all feel more confident than we do now.

Stuart Croasdale, Denver, Colo. (communication to the Secretary*).—I think the New Cornelia ores at Ajo may be considered as typical of all copper ores in the United States that are amenable to acid leaching in a raw condition, since all known ores of this kind contain a certain amount of soluble iron oxides and alumina which will be a contending factor in electrolytic precipitation of the copper.

It is interesting to observe that the results obtained by the authors have verified in almost every instance the predictions made before they started, yet their work has been so efficiently and thoroughly done that it forms a most valuable contribution to the hydrometallurgy of copper, and particularly to the section on electrolytic precipitation. They are to be congratulated on having such an opportunity to definitely advance our knowledge so much in this direction.

I regret that one thing more was not tried, at least on a laboratory scale, and that is, roasting the ore before leaching. In the Ajo ores the

* Received Sept. 29, 1916.

iron exists in all forms of oxidation from chalcopyrite to ferric oxide. The processes of nature have been so gentle that much of this iron is as easily soluble as freshly precipitated hydroxide. Aluminous compounds in the rock also exist in all stages of kaolinization and are likewise soluble in dilute acids. A slight roast would oxidize all of the iron, including the residual pyrite, to ferric oxide and would dehydrate the aluminous compounds and salts to the oxide. Since ignited ferric and aluminum oxides are not readily soluble in dilute acid, it would seem that the lixivium obtained from roasted ore would be practically free from deleterious salts, which would obviate the necessity of depolarizing and "bleeding." Assuming this to be true and the solubility of the copper oxide not diminished, the cost of roasting would be compensated by the recovery of the copper now held in the tailings as sulphide, which is likely to be in ever increasing quantity, and as soluble copper retained by the argillaceous material; also by the expense of roasting pyrite to produce sulphur dioxide, and by the expense of precipitating part of the copper on scrap iron at each cycle of the lixiviant.

Possibilities in the Wet Treatment of Copper Concentrates

BY LAWRENCE ADDICKS,* B S., NEW YORK, N. Y

(Arizona Meeting, September, 1916)

AT the San Francisco meeting of the Institute last year, I presented, through the courtesy of Dr. James Douglas, some results of experiments on the roasting and leaching of concentrator tailings. After it became apparent that flotation rather than leaching was clearly the better method of handling the particular problem under consideration, before dismantling the experimental equipment, some data were secured on roasting and leaching the concentrate itself in competition with smelting. Some of the results obtained are given in the following paper and they are of particular interest at this time in view of the large quantities of flotation concentrates with their somewhat difficult smelting characteristics now being produced.

A complete wet process† consists of roasting and leaching the calcines in dilute sulphuric acid produced from the roaster gases, roasting the residue with salt and leaching with dilute tower liquors (the well-known Longmaid-Henderson process) and recovering the copper, silver and gold by cementation or electrolysis or a combination of both. It is evident, however, that the residue from the first leaching, carrying about 20 per cent. of the copper and all of the silver and gold, can be smelted if preferable. In considering the application of the scheme to individual cases, it must be remembered that freight plays a large part in any reduction process wherein smelting is not conducted at the mouth of the mine, and that it is not practicable today to build small smelting plants for individual operations.

The experiments may be grouped under four main heads: Roasting, leaching, chloridizing residue, and recovery of copper from solutions. The products of two concentrators were used: The Nacozari concentrates were the product of a large modern mill not using flotation, the copper mineral being largely chalcopyrite; and the Tyrone concentrates, the product of an experimental mill including flotation, the copper mineral being chiefly chalcocite. Typical analyses are given in Table 1.

Fig. 1 shows screen analyses of calcines obtained by dead roasting the

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* Consulting Engineer.

† Patent applied for.

TABLE 1.—TYPICAL ANALYSES OF CONCENTRATES USED IN TESTS

	Nacozari	Tyrone
Copper, per cent	14 0	14.0
Silver, ounces per ton	4 0	0.5
Gold, ounces per ton	0.01	Trace
Iron, per cent	31.0	28.0
Sulphur, per cent.	34 0	30.0
Silica, per cent.	13.0	
Alumina, per cent.	3 0	
Lime, per cent	0 6	

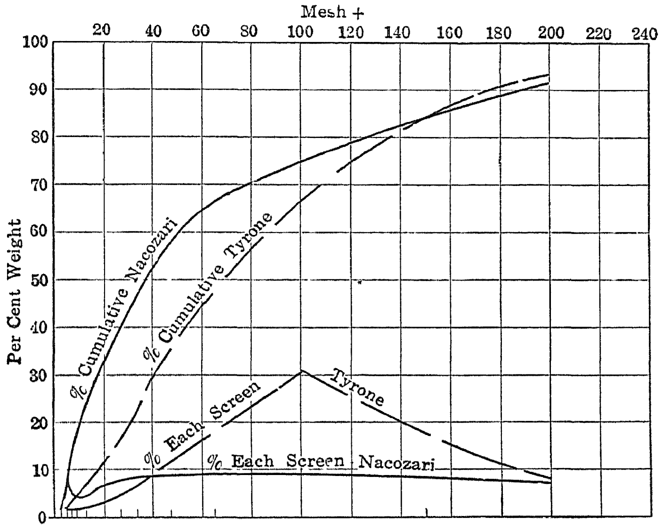

FIG. 1.—SCREEN ANALYSES OF CONCENTRATES CALCINES.

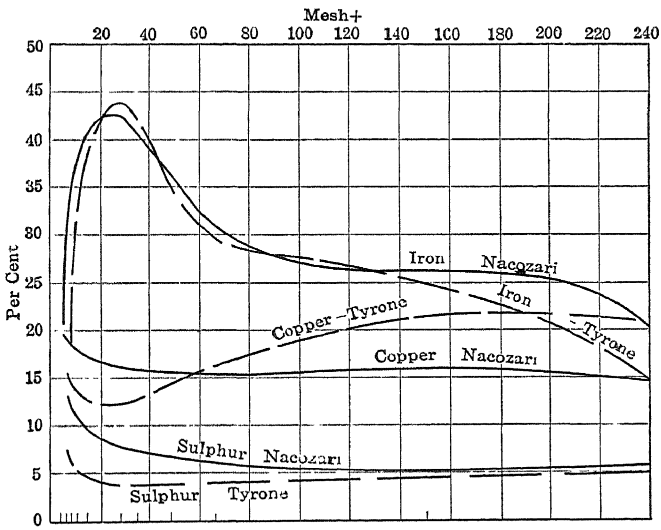

FIG. 2.—VARIATION OF COMPOSITION WITH SIZE OF PARTICLE.

two concentrates as described later. Fig. 2 shows the analysis of the calcines for copper, sulphur and iron for each screen size. The Nacozari concentrates carry considerable coarse jig product while the Tyrone material comes from an ore where the values are finely disseminated and the quantity of 100-mesh is quite marked. The presence of the flotation concentrates in the Tyrone material brings up the copper contents of the fine sizes.

Roasting

The object in roasting is to make as much of the copper and as little of the iron as possible soluble in dilute sulphuric acid. The work is similar to roasting pyrites fines in sulphuric acid manufacture, except that this solubility ratio rather than the complete utilization of sulphur is the

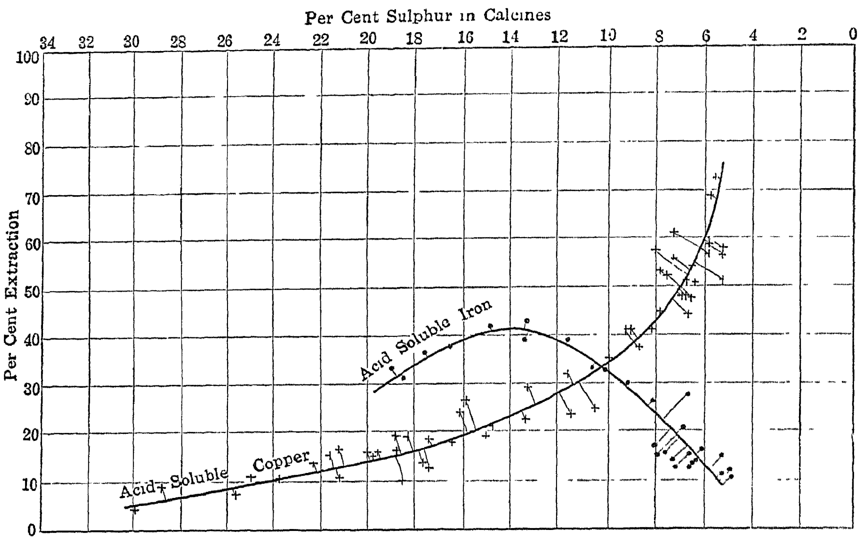

FIG. 3.—NACOZARI CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH McDOUGALL FURNACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

controlling factor. Small-scale work is not very satisfactory as a guide to possible results as it is practically impossible to prevent overheating due to rapid oxidation of sulphur in a laboratory experiment. An 18-ft., water-cooled, six-hearth McDougall was used, the speed of rotation being gradually cut down until dead roasting conditions were obtained. Greater tonnages could doubtless have been obtained in a seven-hearth furnace.

Many samples were taken from various hearths and the acid-soluble copper and iron determined by leaching with 4 per cent. H_2SO_4 in the laboratory. The results of these tests are given in Figs. 3 and 4. The

results of tests on a series of sixth-hearth samples to determine the relation between tonnage and sulphur elimination are plotted in Fig. 5. It is

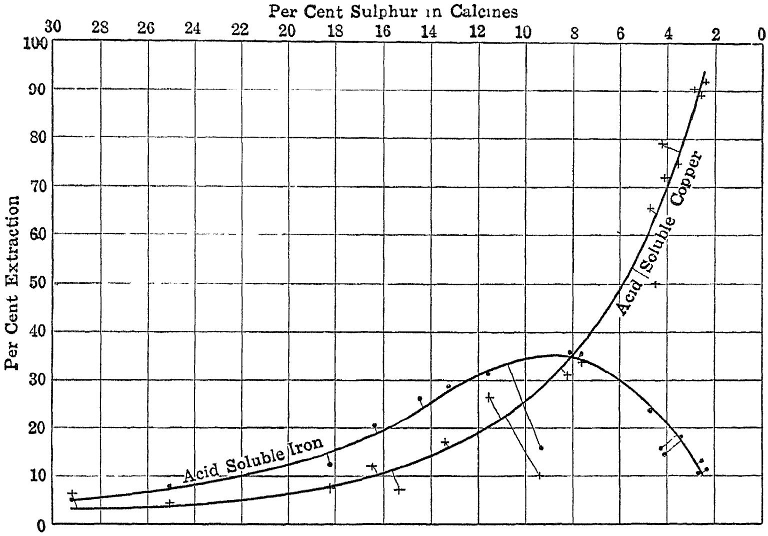

FIG. 4.—TYRONE CONCENTRATES ROASTED IN 18-FT. SIX-HEARTH MCDOUGALL FURNACE AND LEACHED IN 4 PER CENT. SULPHURIC ACID IN LABORATORY.

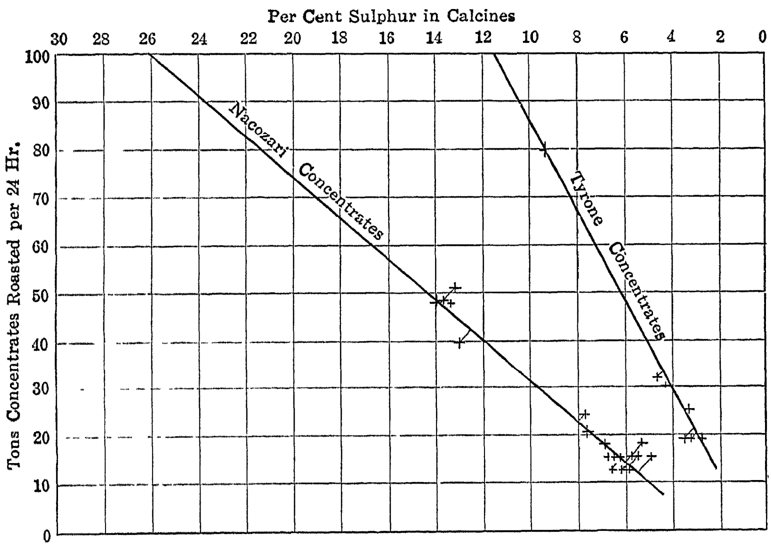

FIG. 5.—ROASTER CAPACITY VS. ELIMINATION OF SULPHUR.

evident that the chalcocite can be oxidized much more readily than the chalcopyrite, although size of particles has something to do with this. An investigation of the solubilities of the various sizes of particle was carried

out by screening some of the calcines, as shown in Fig. 6. As would be expected, the finer particles are the more thoroughly oxidized; the jig product in the Nacozari concentrate is one reason for the poorer results obtained in the treatment of this material.

In general, these large-scale experiments indicate the possibility of reasonably obtaining the results desired—high copper and low iron solubility—but it is obvious that the residue after leaching will contain sufficient copper values to require retreatment, aside from the fact that any silver and gold will remain in this residue.

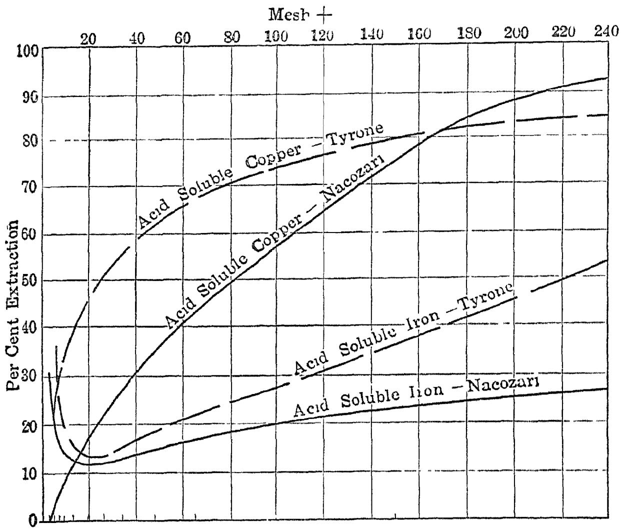

FIG. 6.—RELATIVE SOLUBILITIES OF VARIOUS SIZED PARTICLES OF CALCINE.

Leaching

As shown in the paper presented last year, such satisfactory results in the extraction of copper values from tailings were obtained by dumping the hot calcines from the furnace into a leaching trough, the few seconds' agitation thus obtained extracting almost as much as prolonged treatment in other apparatus, that the same idea was tried out with the concentrate calcines. It was not possible, for various reasons, to handle the output of the furnace directly, so the calcines were stored and then fed to a bucket elevator which in turn delivered into a V-trough in which the leaching liquor was flowing. The results were here disappointing, as although there was instant extraction of perhaps half of the soluble copper, a prolongation of the trough to give 60 sec. travel did not greatly increase this amount. It was definitely shown in the laboratory as well that prolonged agitation was necessary to extract all of the soluble copper. The

leaching trough delivered into an acid-proof drag consisting of an endless belt with angle rakes, of the type commonly used in concentrators for dewatering. This acted more or less as a classifier, the very fine residues being carried over with the liquor, from which they were subsequently separated by settling. As this still gave insufficient agitation to the sands, a Parral tank was tried, but it was found that they were too heavy to yield readily to any sort of air-lift agitation. A Dorr classifier was then added to the apparatus and this did better. It was found, however, that it was necessary to pass the residues six or seven times through the leaching process in order to obtain an extraction equal to that shown by

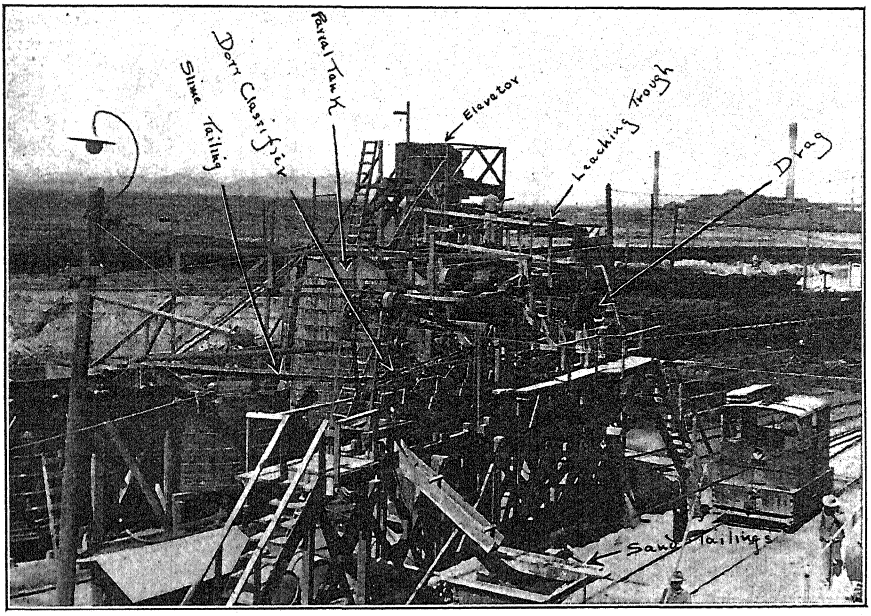

FIG. 7.—EXPERIMENTAL LEACHING PLANT.

laboratory tests on the calcines. Fig. 7 gives a general view of the leaching apparatus.

The large-scale leaching tests were confined to the Tyrone material, a lot of 30 tons of calcines from some of the roasting tests being used. The first runs on a lot of 17 tons of not quite dead-roasted material running 4 per cent. sulphur gave results that were satisfactory except in that too much iron was dissolved, causing a needless consumption of acid and embarrassing any electrolytic scheme of recovery. Later, better-roasted material was available and a careful record kept of the metal balance and acid consumption, with the results shown in Table 2.

These figures check reasonably close except in the case of iron; but it must be remembered that various iron parts in the apparatus used were

TABLE 2

Time Through	Per Cent. Cu in Tails	Trough Fed	Dorr Fed
1st	7.7	Acid	Water
2d	6.0	Acid	Water
3d	5.2	Acid	Water
4th	3.6	Acid	Acid
5th	3 3	Acid	Water

Extraction by Heads vs. Tails

	Weight, Pounds	Copper		Iron		Alumina	
		Per Cent.	Pounds	Per Cent.	Pounds	Per Cent.	Pounds
Heads	8,360	15.48	1,292	31.00	2,590	5.60	468
Tails	5,600	3 50	196	43 52	2,440	7.02	393
Extraction	.....	84.70	1,096	5.80	150	16.00	75
Extraction per lb. of Cu.	.....	...	1.00	...	0 14	..	0 07

Extraction by Analysis of Liquors

	Weight, Pounds	Copper		Iron		Alumina	
		Per Cent.	Pounds	Per Cent	Pounds	Per Cent	Pounds
Heads	51,538	0.46	238	0.20	104	0 47	246
Tails	86,910	1.46	1,271	0.76	657	0.45	395
Extraction	......	80.00	1,033	21.30	553	31.90	149
Extraction per lb. of Cu.	......	...	1.00	...	0.53	...	0.14

attacked by the liquor, which would artificially increase the iron taken into solution.

The acid consumption was 2,495 lb. of 100 per cent. H_2SO_4 for the run. This is equivalent to 2.28 lb. per pound of copper extracted. Laboratory tests on the same calcines indicated 2.0 lb. The leaching was done at about 125° F. with 5.6 per cent. free acid in the liquor entering the trough.

In general, when a 15 per cent. copper calcine is fed to the trough, the residue at the end of the trough will run about 8 per cent. Cu, the extraction representing the instantaneously soluble copper. This residue can be brought down to about 3.5 per cent. Cu by suitable agitating means, with a consumption of a little over 2 lb. of acid per pound of copper, and with the extraction of but little iron. The final residue weighs only about 60 per cent. of the original concentrate before roasting.

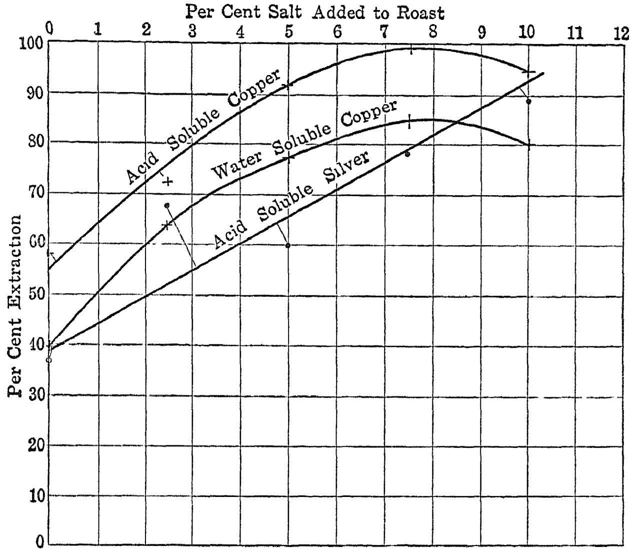

FIG. 8.—CHLORIDIZING LEACHED CONCENTRATES CALCINES.

Roasted 1½ Hr. at 975° F. with Addition of Salt and 10 Per Cent. Raw Concentrates.
Calcines: 5.6 Per Cent. Copper, 1 9 Oz. Silver and 2 5 Per Cent. Sulphur.
Raw Concentrates: 14 4 Per Cent. Copper, 0.55 Oz. Silver and 34 Per Cent. Sulphur.
Liquor: 5 Per Cent. Na_2SO_4, 5 Per Cent Na Cl, 5 Per Cent. Fe Cl_2 and 0.5 Per Cent. HCl + H_2SO_4.

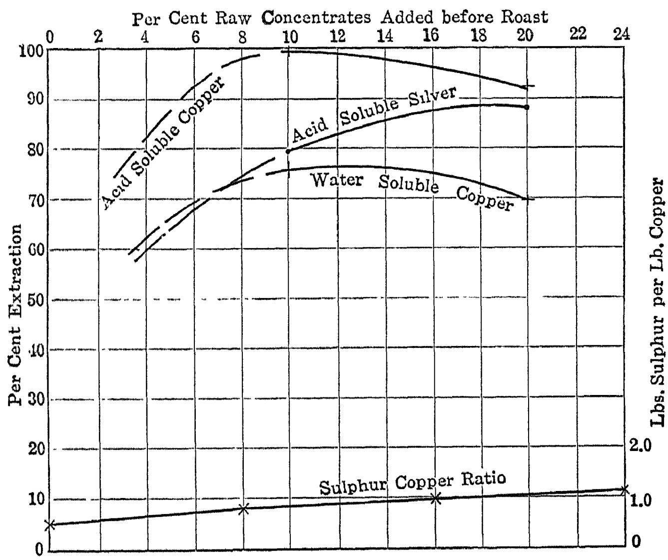

FIG. 9.—EFFECT OF SULPHUR-COPPER RATIO UPON EXTRACTION. 7½ PER CENT SALT ADDED TO ROASTING "MIX".

Chloridizing Residue

No large-scale work was done on the chloridizing of the residues from the first leaching. The analysis of these residues, however, differs from that of pyrites cinder, so long successfully treated by this process, only in the amount of silica present. Various small-scale experiments were tried and 50 lb. or so were sent for test to a plant where the Longmaid-Henderson process was in operation. Both sets of experiments were entirely satisfactory.

A small lot of leached residues was prepared for test. These contained 5.6 per cent. Cu, 1.9 oz. Ag, and 2.5 per cent. S. Raw concentrates for adjusting the sulphur-copper ratio were used, containing 14.4 per cent. Cu, 0.55 oz. Ag, and 34 per cent. S. Fig. 8 shows the extractions with varying percentages of common salt added to the "mix" after roasting in an electric muffle furnace 1½ hr. at 975° F. and leaching in a liquor carrying 5 per cent. Na_2SO_4, 5 per cent. NaCl, 5 per cent. $FeCl_2$, and 0.5 per cent. HCl + H_2SO_4. Fig. 9 shows the effect of varying the sulphur ratio. The results show a 99 per cent. copper and a 79 per cent. silver extraction. The report on the lot of residues sent away fully confirmed these results.

Recovery of Copper from Solutions

The liquor from the chloridizing plant would doubtless be reduced to argentiferous copper cement by iron. But 20 per cent. of the original copper is involved. The sulphate liquor from the first leach could be precipitated on iron if desired, or with certain limitations would be suitable for electrolytic deposition of the copper and regeneration of the acid. Sulphide concentrates carry from 1 to 2 lb. of sulphur per pound of copper, equivalent to from 3 to 6 lb. of 100 per cent. H_2SO_4, less process losses, if the roaster gas is oxidized to sulphuric acid. Since the leaching calls for but a little over 2 lb. of acid per pound of copper, plus tailings losses, it would seem possible, therefore, to figure on a simple cementation plant, considering electrolysis as a competitor on a basis of relative profit and not of necessity.

Discussion

F. N. Flynn, Clifton, Ariz.—As a number of my associates in Arizona know, for a great many years, I have felt that leaching was one of the coming problems. We are about to start experiments at Clifton, along the idea of leaching wet flotation concentrates direct, without roasting—differing in this respect from Mr. Addicks' suggestion. We are prompted in starting these investigations by the difficulty in transporting and handling flotation concentrates, and consideration for drying and

roasting dust losses. I have had the idea for 5 or 6 years past, that those very fine concentrates could best be treated by leaching at the concentrating plant, thus eliminating our transportation difficulties to the smelter, and dust losses at the smelter. Given flotation concentrate, if it must be dried and roasted, it would, in our case, undoubtedly be cheaper to smelt the calcine, but if it can be leached raw and wet and the drying difficulties and dust loss eliminated, it would be very advantageous to leach it at the mill. I realize that it will be a difficult problem, and one which may take several years to develop, but if developed, it would pay for many years' expense of investigation, and we have been encouraged by our general manager to investigate its merits.

The 2,000-Ton Leaching Plant at Anaconda

BY FREDERICK LAIST,* B. S., AND HAROLD W. ALDRICH,† M. E., ANACONDA, MONT.

(Arizona Meeting, September, 1916)

AFTER a series of experiments covering a period of about three years, extending from the spring of 1912 to the spring of 1915, a 2,000-ton leaching plant for the treatment of the accumulated copper concentrator tailing was built and put into operation. During the experimental period, the first step was that of laboratory experiments or beaker leaches. The results on this small scale were so satisfactory that a

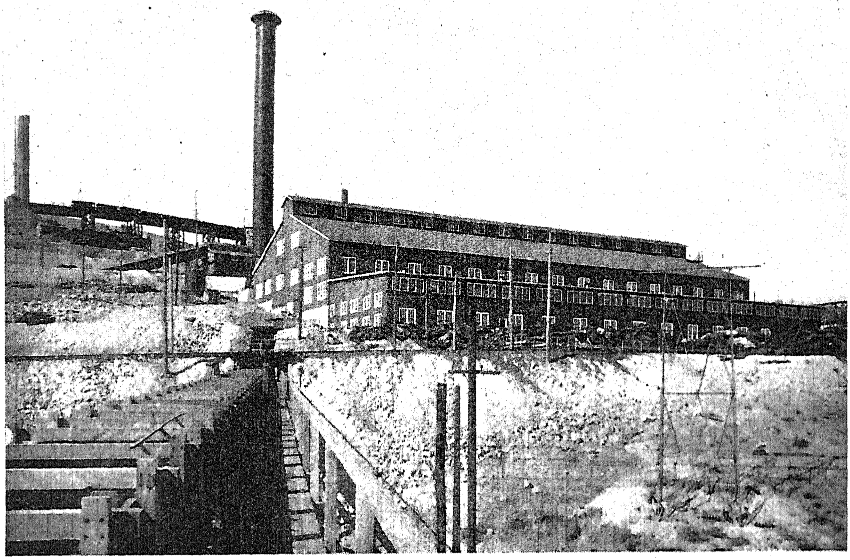

FIG. 1.—LEACHING PLANT AT ANACONDA.

small operating plant, capable of handling 5 tons of roasted tailing per day, was installed. Again, the results proved satisfactory and an 80-ton plant was built and operated continuously from August until February, 1913. In this plant, full-sized roasting and leaching units were

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† Superintendent of Leaching Plant, Anaconda Copper Mining Co.

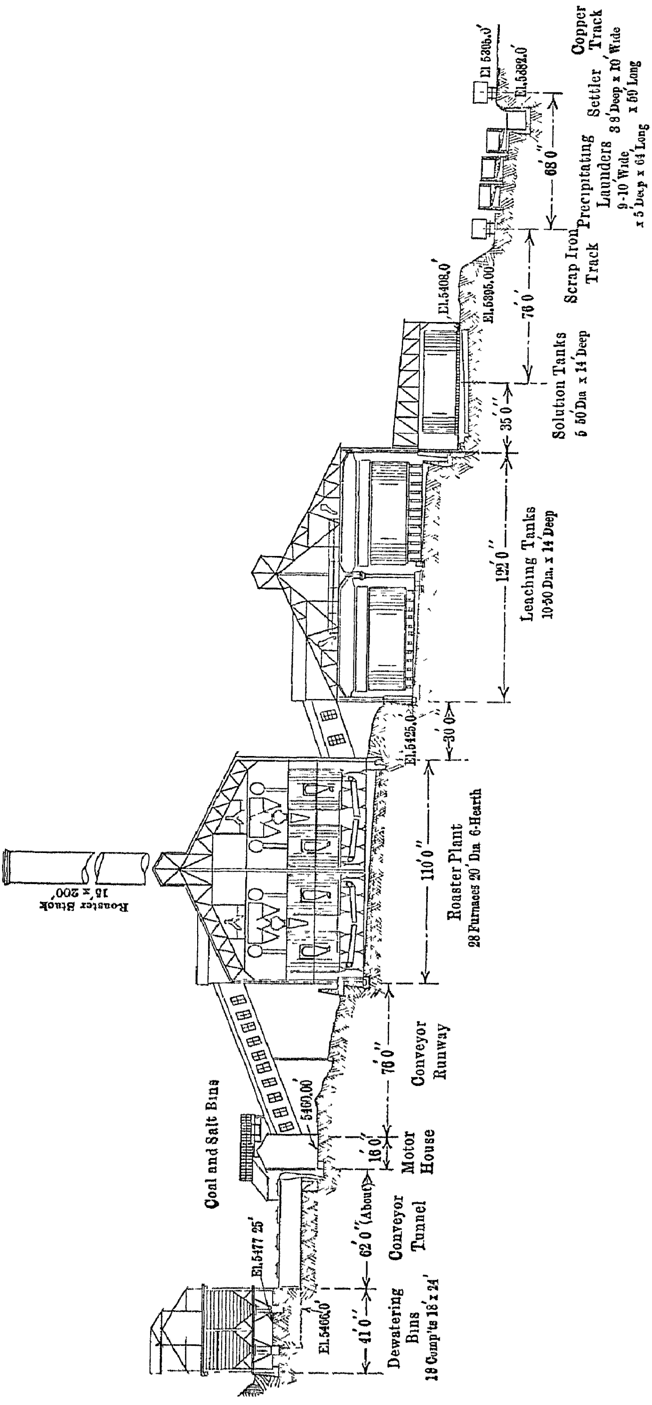

FIG. 2.—GENERAL PROFILE OF 2,000-TON LEACHING PLANT AT ANACONDA.

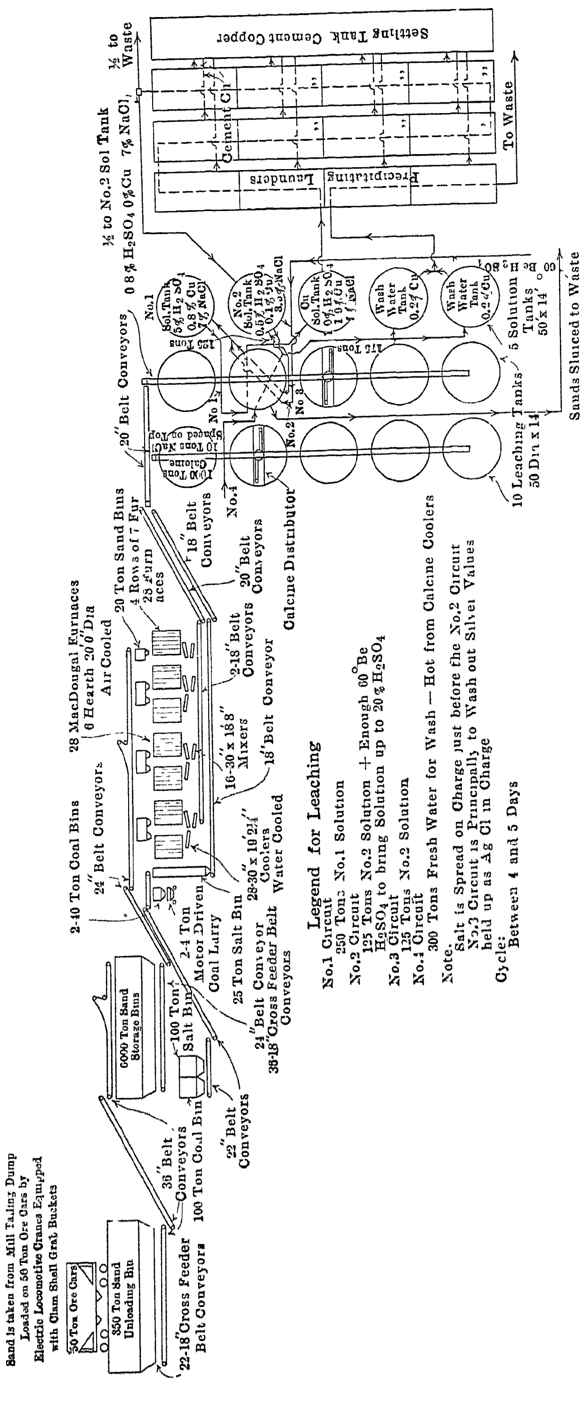

FIG. 3.—FLOW SHEET OF LEACHING PLANT.

used. The results obtained by the operation of the 80-ton plant proved that the roasting and leaching of deslimed concentrator tailing could be profitably done, so in the early spring of 1914, the construction of a 2,000-ton leaching plant was begun (Figs. 1 and 2). Operation of the plant began on May 13, 1915. A flow sheet of the plant is shown in Fig. 3.

Leaching Plant Feed

The accumulated tailing in the New Works dump is estimated at about 20,000,000 tons. The dump consists of the concentrator tailing discharge, over a period extending from February, 1902, to the present date. According to daily samples taken during that period, the copper content is about 0.64 per cent. and the silver 0.48 oz. per ton. The peak of the dump, or the point where the tailing is discharged, carries about 0.75 per cent. copper, while down toward the toe, and where present excavation is taking place, the copper content is only 0.57 per cent. About 3 lb. per ton of the copper is oxidized, the remainder being sulphide.

The following shows an average screen analysis of the tailing in the dump:

Cumulative Per Cent.

+20 0	+40.0	+60.0	+80.0	+100.0	+160.0	+200 0	−200.0
22.6	64 7	82.0	89.1	93.8	97.7	98 5	1.5

An average analysis of tailing being treated at the present time follows:

H_2O Per Cent.	Cu Per Cent.	Ag Ounces	SiO_2 Per Cent.	FeO Per Cent.	S Per Cent.	Al_2O_3 Per Cent.	CaO Per Cent.
5.63	0.573	0.487	81.0	2.8	1.9	9.8	0.4

The dump is being excavated by a Bay City Industrial Works electric hoist, equipped with a 4-yd. bucket. The hoist loads the tailing into 50-ton ore cars. These are hauled to the leaching plant storage bins in trains of 15 cars each. The hoist is capable of hauling 3,000 tons in 8 hr.

Conveying and Storage Equipment

Unloading Pit

The loaded cars of tailing are spotted by means of a 25-ton electric locomotive operated by the third-rail system. The unloading pit is a steel bin capable of holding 350 tons and is of sufficient length to allow three cars to be unloaded at a time. On top of the pit is a 2-in. grizzly, through which all tailing must pass before going to the storage bins. This grizzly serves to keep out rocks or lumps which would block the

feeders, furnaces, etc. Under the steel bin are 22 short-belt feeders, each feeding from its own gate and running at right angles to the length of the bin. These feeders discharge onto a 36-in. belt running lengthwise, which in turn delivers the sand to another 36-in. belt traveling on an 18° angle to the top of the storage bins. The system will handle over 3,000 tons of tailing in 8 hr.

Storage Bins

The storage bins will hold 6,000 tons of sand tailing. This gives between two and three days' supply for the leaching plant, in case of railroad troubles or difficulties due to cold weather. The bins are of substantial wood construction and are inclosed, the walls and roof of the building consisting of wood sheathing covered with corrugated iron. The bins are arranged in a double row and are hopper-bottomed. The tailing is distributed over the bins by means of a 36-in. belt and movable tripper.

Underneath are 36 gates, each with its hopper from which an 18-in. belt feeder delivers to a 24-in. belt running the full length of the building, in the center. By means of other belts, traveling through a tunnel under the railroad tracks, and up an 18° incline, the feed is delivered to the top of the furnace building.

There are also small coal and salt storage bins from which a 22-in. belt system conveys coal and salt to bins in the roaster building, the salt being drawn from there to the leaching building as it is required. All belts run at approximately 400 ft. per minute.

Roasting Equipment

The furnace building is 232 by 110 ft. and is of steel and concrete construction (Fig. 4).

There are 28 McDougall-type, six-hearth furnaces arranged in four rows of seven each. The furnaces are 20 ft. in diameter, each being equipped with two fireboxes, diametrically opposite, the flame entering over a fire bridge directly into the third hearth, the top hearth being designated as the first. The grate dimensions of each firebox are 3 by 4 ft. (Figs. 5 and 6).

Each furnace has a 20-ton feed hopper, to which the tailing is delivered by means of two 24-in. belts each equipped with a movable tripper. The furnaces are fed by 14-in. belt feeders drawing from these hoppers, the amount of feed being controlled by gates which are operated by means of a screw adjustment, the feed dropping through a hole in the top arch, directly onto the top floor (Fig. 7). During the time the feed is on the two upper floors, it is dried and heated; as it drops

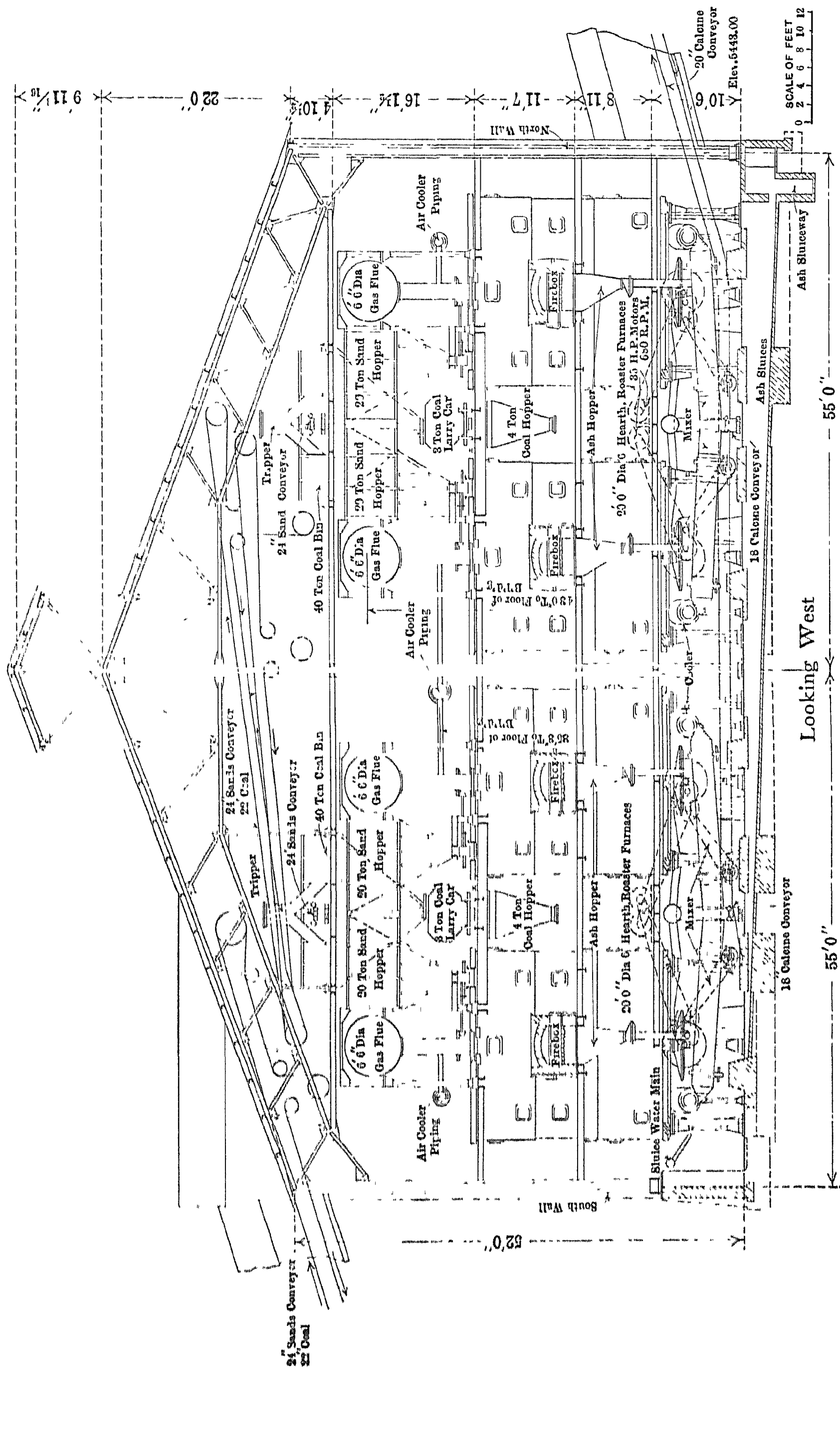

FIG. 4.—CROSS-SECTION THROUGH ROASTING PLANT.

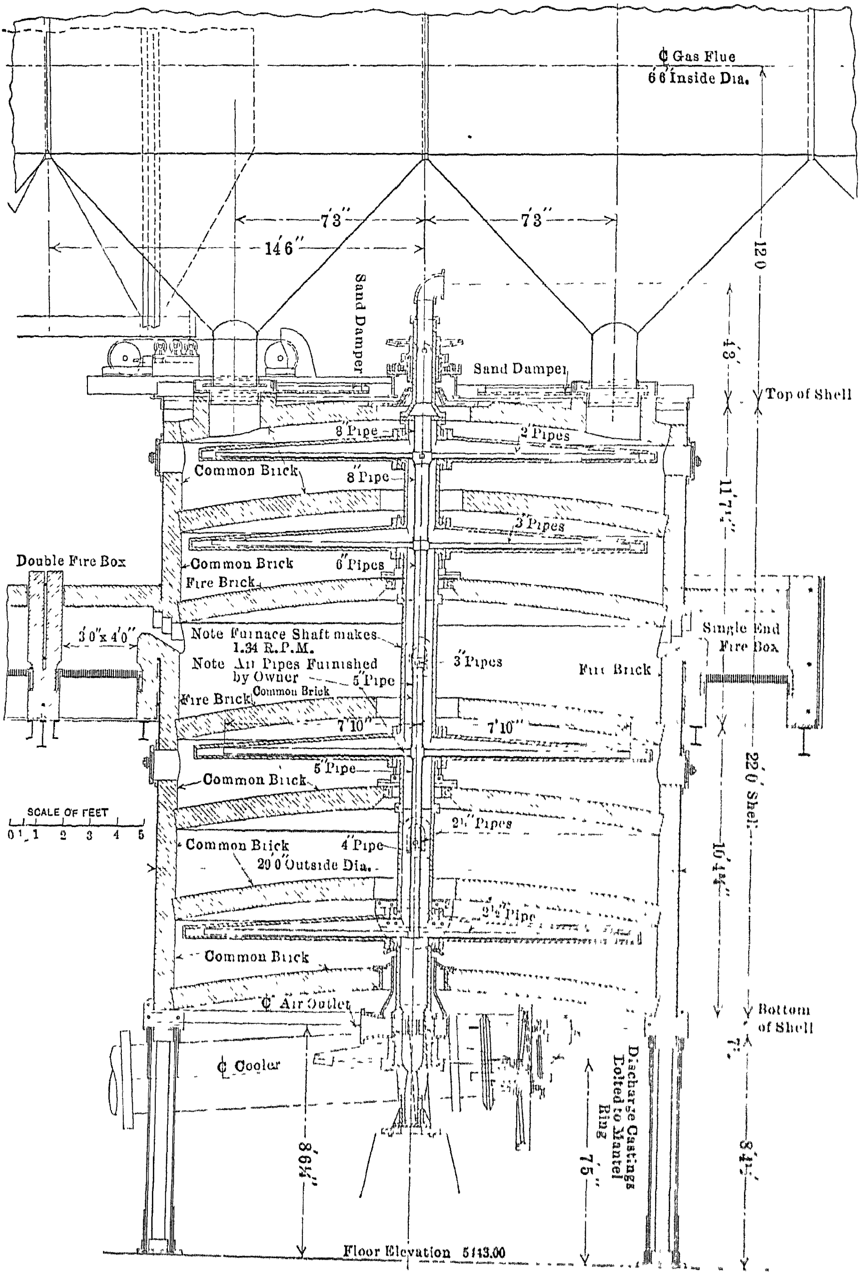

FIG. 5.—VERTICAL SECTION THROUGH MCDOUGALL FURNACE.

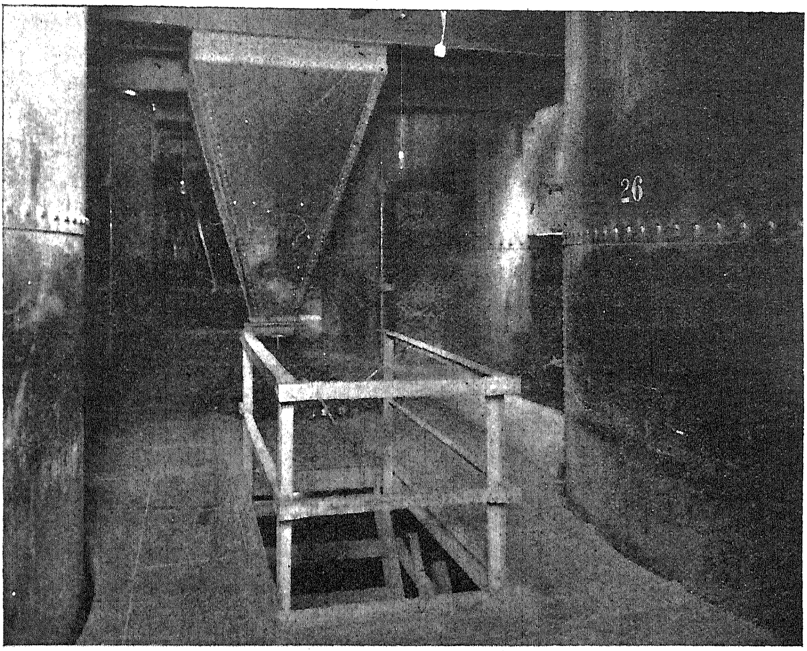

Fig. 6.—Firing Floor of Roaster Plant

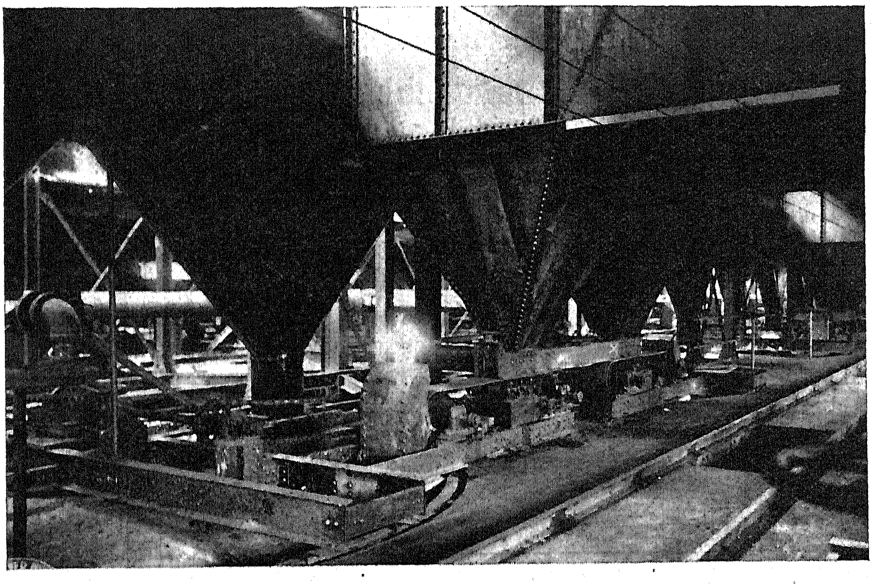

Fig. 7.—Feed Floor of Roaster Showing Feed Hoppers, Flue Connections and Feeding Mechanism.

to the third or fired floor, the sulphur ignites. The three lower floors are kept hot by the combustion of the sulphur in the tailing.

Four flues run the length of the building, one over each row of seven furnaces. Each furnace has two opposite connections, from the top hearth to the flue. All four flues lead into a balloon flue with a downtake of 45°. The balloon flue enters the 15 by 200-ft. steel stack with a 45° uptake. The stack is lined with brick. In the bottom of the balloon flue is a 6-in. screw conveyor which delivers the flue dust to the belt-conveyor system which receives the calcine from the furnaces.

The furnaces are air-cooled, the air being furnished by four No. 11 Buffalo blowers each direct-connected to a 50-hp. motor. The air intake is at the top of the furnace shaft and the discharge at the bottom. The hot air does not enter the furnaces, but is delivered to the leaching and solution buildings for heating purposes by a suitable piping system. When it is not needed for this it is discharged into the atmosphere.

Each furnace is equipped with a cylindrical cooler, 30 in. in diameter and 19 ft. long. The cooled calcine enters a mixer or concrete-lined steel cylinder, at the head end of which a very small stream of water is added to settle the dust. The mixer discharges a moist warm calcine to an 18-in. conveyor belt, and by a system of conveyors the calcine is delivered to the leaching building.

The ashes from the fireboxes drop into launders and are sluiced out through the main tailrace.

Leaching Plant Equipment

The leaching building is 293 by 122 ft. and is of steel and wood construction. It contains 10 redwood tanks each 50 ft. in diameter, and 14 ft. deep, lined with 8 lb. lead. The average charge to a tank is about 1,000 tons of calcine.

The tanks are equipped with an ordinary filter bottom, made of 1¼-in. slats resting on 2 by 4-in. pieces. Above this are two layers of heavy cocoa matting and on top of the matting is a grating, made of 1¾ by 3½-in. material, with 6-in. square spaces. The grating fills with calcine 3½-in. deep and serves to keep the force of the sluicing water from tearing the matting. The acid solutions rot the cocoa matting, but if not disturbed, it will hold its shape and be an efficient filtering medium long after it is too much decomposed to handle.

The tanks are in two rows of five each. A 20-in. conveyor belt travels over each row, and, by means of a tripper, the calcine is dumped into a distributor, which spreads it over the tank (Fig. 8).

Each tank has three lead pipes 4 in. in diameter and one 4-in. iron pipe entering at the top. The lead connections are for strong and dilute acid solutions and the iron pipe is for wash water. Above the level of the

leaching tanks an iron storage tank is provided for holding the stock of concentrated acid. Its capacity is about 120 tons of 60° Bé. acid. All concentrated acid, used to raise the acid strength of any solution, is added to the solution as it goes on the charge in the leaching tanks.

There are seven 10-in. sluicing gates in the bottom of each leaching tank, one in the center and six spaced equidistant from each other in a circle about half way between the center and the circumference of the tank. These discharge into launders which connect with the main tailings launder.

"Acimet" valves and lead piping are used throughout for handling dilute and concentrated acid solutions.

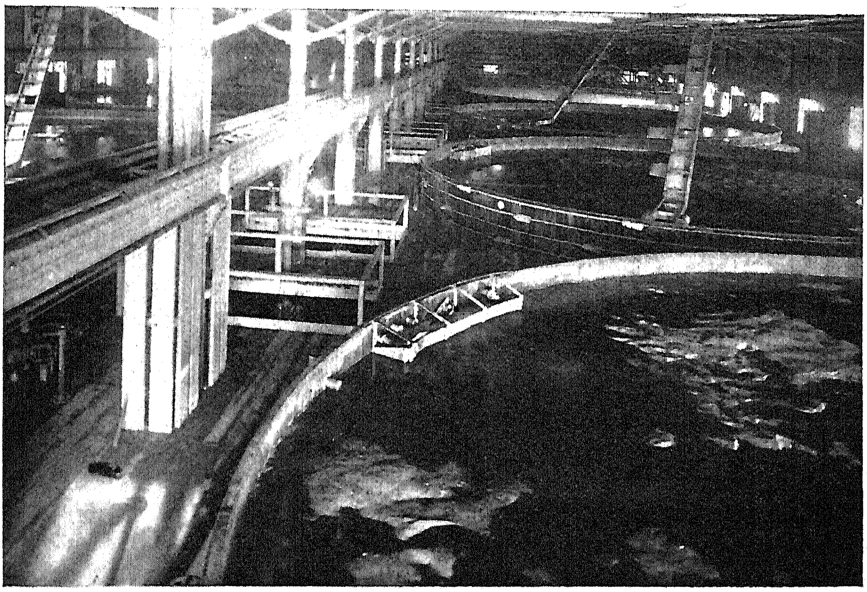

FIG. 8.—VIEW OF LEACHING TANKS AND DISTRIBUTING CHUTES.

The floors of both the leaching and solution buildings are of concrete, and are painted with an asphalt-tar mixture for acid proofing. These floors slope to a gutter which connects to a pump sump and in this way any overflow or leakage of solution is saved and returned to the system.

The solution tank building is a leanto off the leaching building and contains five solution-storage tanks. These are 50 ft. in diameter and 14 ft. deep. Solutions drain from the leaching tanks to the storage tanks and are pumped to the top of the leaching tanks, from the solution tanks, by means of vertical shaft, direct-connected, hard-lead, centrifugal pumps.

Precipitating Division

The precipitation of the copper and silver is accomplished with scrap iron. The precipitating launders are of concrete, each about 250 ft. long and having a section of 4 by 8 ft. available for containing iron. Each launder is partitioned off into four sections by concrete walls. Any of the 12 sections may be bypassed for the purpose of cleaning up. In the bottom of the launders is a heavy wood grating, upon which the iron rests, leaving a space about 6 in. under it, for accumulation of any cement copper which may drop off the iron. In the side of each section, at the bottom, are four 6-in. holes, toward which the concrete bottom slopes. These holes discharge into launders which carry the copper to a settling tank. There it is washed and stored and finally excavated with a clam-shell bucket, loaded into standard railroad cars and, at present, shipped to the briquetting plant before blast-furnace treatment. An electrically operated Brown hoist, equipped with a lifting magnet, handles the scrap iron and loads the copper.

Details of Operation

Roasting

The tailing is subjected to a simple oxidizing roast, no particular care being taken to obtain a large amount of sulphate. The sulphur content of the feed is about 2.2 per cent. and that of the calcine about 0.6 per cent. One-third of the total sulphur in the calcine is in the form of sulphate. When too hot a roast is attempted in order to decrease the total sulphur content, a certain portion of the copper is rendered insoluble in all ordinary acids, with the exception of hydrofluoric.

A pyrometer is inserted over the fourth floor of every furnace, and by means of these, the firemen are able to keep constant control of the temperature. The best results are obtained by keeping the fourth floor at about 500° C. The hottest hearth in the furnace is the third or fired floor, and averages about 535° C.

The water for the calcine cooler system enters at about 40° C., is discharged at 65° C., and is piped to the solution building, where it is used to heat the circulating solutions. It is then pumped back and used again in the coolers. The calcine after passing through the coolers, and after the addition of 1 per cent. moisture, while going through the mixers, has a temperature of about 45° C. During the conveying from the roasters to the leaching tanks, this temperature is lowered to 40° C.

Leaching

The leaching is done by continuous downward percolation, no circulation or upward percolation being used. The percolation rate will vary

from 3 in. per hour with the first solution to as high as 10 in. per hour with the wash water. As nearly as possible, all solutions and wash waters go on the charge at 40° to 50° C.

It requires about one-fourth of the weight of calcine, in weight of solution, to saturate a charge thoroughly.

There are five solution tanks: One for storage of No. 1 solution, one for No. 2 solution, one for copper solution, and two for wash-water.

	Cu Per Cent	H_2SO_4 Per Cent.	NaCl Per Cent.
No 1 solution tank. .. .	0.8	5 0	7.0
No. 2 solution tank...... . ..	0.1	0.5	3 5
Copper-solution tank	1.9	1 0	7 0
Wash-water tanks	0 2	1 0	1 0

After a tank is charged with calcine and leveled, 250 tons of No. 1 solution is added as fast as the charge will absorb it. The drain valve is always open, so, as soon as the solution reaches the bottom of the tank, it commences to drain to the copper-solution storage tank as copper solution. From the copper-solution tank there is only one outlet, which is to the precipitation launders. After traveling through the launders, two-thirds of the solution is returned to the No. 2 solution tank and the balance wasted. This waste is necessary to keep impurities such as iron and aluminum sulphates from building up in the system.

When the No. 1 solution has all been added to the leaching tank, the solution is allowed to drain until none shows on top of the calcine, when 1 per cent. of the weight of the charge, of common salt (NaCl), is spread over the calcine. On top of the salt is then added 100 tons of solution from No. 2 solution tank, but with additional strong acid to bring it to 20 per cent. H_2SO_4. Following the 20 per cent. acid solution, 150 tons of No. 2 solution is added, but without additional strong H_2SO_4. This scheme gives a zone 4 or 5 ft. in depth of very strong chloridizing solution, traveling down through the charge. There is about 8 per cent. of ferrous and ferric iron in solution, which, with the salt, forms ferric chloride, in itself a very corrosive reagent, even dissolving a considerable amount of unroasted sulphide. This chloridizing action also gives the silver extraction, as without it very little silver is recovered. The 150 tons of No. 2 solution which follows the 20 per cent. acid is for the purpose of washing out silver chloride and dissolved copper which may have been held in the calcine. It carries very little copper or acid, but is fairly high in salt content, and therefore better than a clean water wash. Following the last acid solution, about 300 tons of hot, clean water is added.

The two portions of No. 2 solution, one at 20 per cent. acid, and the other at 0.5 per cent. acid, after percolating through the charge, drain to the No. 1 solution tank.

The wash water, less a quantity sufficient to make up for the discarded solution, drains to the two wash-water tanks. The balance goes to the No. 2 solution tank and adds enough to make up the amount of solution discarded from the precipitating division each day.

Precipitating

The practice here is too old to necessitate much explanation. The main advantages in the practice at this plant over the usual practice are the large launders, which make it possible to put in large and odd-shaped pieces of iron, and the presence of salt in the solutions which prevents the copper from plating on the iron, and makes a soft spongy cement copper which is easily washed off with a hose, leaving the iron clean for more precipitation. It is never necessary to remove the iron for cleaning. The silver is recovered by precipitation on the precipitated copper.

Results

The resulting cement copper carries about 70 per cent. copper. The following data are taken from the reports for the month of October, 1915. This is a representative month, but it is certain that the results will be improved upon, after longer operation.

Sand tailing treated, tons	70,401.00
Copper in feed, per cent	0 575
Silver in feed, ounces per ton	0.45
Copper in tailing, per cent	0.082
Silver in tailing, ounces per ton	0.14
Sulphuric acid (60° Bé.), pounds per ton of feed	64.90
Coal, per cent. of feed	3.30
Salt, per cent. of feed	1.52
Iron, pounds per pound of copper	2.00

The plant during this month made an extraction of about 80 per cent. of the copper and 60 per cent. of the silver. This is less than indicated by assays of heads and tailings owing to various plant losses of which the largest is dust from the roasting furnaces amounting to about 4.5 per cent. of the copper in the feed.

Analyses of Feed and Tailing

	Cu, Per Cent.	Ag, Ounces	SiO_2, Per Cent.	FeO, Per Cent.	S, Per Cent.	Al_2O_3 Per Cent.	CaO, Per Cent.
Feed	0.575	0 45	81.3	3.0	2.1	9.4	0.4
Tailing	0.082	0.14	84.7	2.4	0.4	8.7	0.4

Discussion

F. N. Flynn, Clifton, Ariz.—I would like to ask Mr. Mathewson what percentage of his leaching liquor is wasted at this time? It has a bearing on the question in connection with the New Cornelia, and I think it is a very important question.

E. P. Mathewson, Anaconda, Mont.—I am not sure what the percentage is now, but we don't discard the strong liquors. Something like 25 per cent. of the wash water taken out each day is sent over a special set of scrap iron tanks, and the copper recovered there. We had a little trouble at first. We did not proceed to use the sponge iron for the reason that the plant we originally contemplated was abandoned for flotation. Our original idea was to use a leaching process on all the tailings of the mill, but we found it would take a great deal of time to build a plant; and the cost of operating the plant we estimated to be about the same and the recovery the same. It was a question of time with us, and we decided to adopt flotation. That changed our plans. We now have the leaching plant, treating the tailings of the old dump, and the flotation applied to the current tailings. The sponge iron was figured for the large plant. We can get all the scrap iron necessary for the small plant—plenty from the scrap produced in the main plant of the smelter.

The Chairman (H. W. Morse, Los Angeles, Cal.).—I would like to open this meeting for a little while to the general subject of leaching. We ought not to hold back if we have any new schemes for the future. I know that those of us who are tangled up with this leaching work will appreciate any suggestions that will make us work harder and scheme harder to pull through these past difficulties on leaching.

F. S. Schimerka, Clifton, Ariz.—Under the direction of J. W. Bennie, General Manager of the Shannon Copper Co., Clifton, Ariz., it has been my valued privilege to coöperate in the elaboration of a hydrometallurgical process aiming at the recovery of copper in mill tailing resulting from a gravity concentration of a low-grade semi-oxidized sulphide ore occurring in a porphyritic gangue.

The experimental work which has been carried on with a 25-ton plant is completed, and the company is building at present a leaching plant on a larger scale, the first unit consisting of a 150-ton roasting furnace and accessories for leaching the calcine.

I wish to lay before you only the most essential points of our process, the outstanding feature of which is the non-application of acids or any other chemical in the leaching operation proper. After a sulphatizing roast conducted under well-defined conditions in a mechanical roasting furnace of the multiple-deck type, the calcine is treated with water only.

The separation of the copper liquor will be effected by decantation, and the copper, at least for the present, precipitated by scrap iron. The character of the tailing, which is highly basic and remains so even after roasting, prohibits the application of sulphuric acid, which is consumed to the amount of more than 10 lb. for each pound of copper dissolved from the calcine. Our mill tailing, all of which passes through a 1-mm. opening, contains an average of 20 lb. of copper to the ton, 1 per cent., we may say. Fifty-five per cent. and more of the total copper is present in oxidized condition, mainly as malachite and azurite, the balance is chalcocite. Sulphur is present in the amount of 1 per cent., mostly as pyrite. The precious metals are practically absent, and that no attempt at their recovery need be made has materially simplified our working procedure.

The manner in which the roasting of the raw tailing is conducted is, of course, vital to the process, there being no other means employed to convert most of the copper into water-soluble sulphate than the agencies of heat, oxygen and sulphur during the passage of the material through the furnace. A close regulation of the temperature in the roaster by means of pyrometer control is essential. Tests have shown that the re-formation of water-insoluble basic copper sulphate and oxide takes place when the temperature rises closely to 900°F., and we aim at keeping the temperature on the hottest floor between 830 and 860°F. We have encountered no difficulty in accomplishing this. It is, however, necessary to limit the amount of sulphur in the charge to a quantity which in an empirical way has been found to produce the best results, and which excludes the possibility of local overheating, as would take place by the oxidation of a large amount of pyrite. Moreover, any excess sulphur will, by formation of sulphur dioxide, decrease considerably the oxidizing effect of the hot gases striking through the furnace. Our roaster gases contain less than 0.1 per cent. of sulphur dioxide when issuing from the stack. For these two reasons, we aim at keeping the charge as poor in sulphur as we can afford to do without injurious effect upon results. Taking the copper contents in the tailing as a basis, we adjust the sulphur in the charge to 1½ lb. for each pound of copper in the tailing. This necessitates the addition of ½ per cent. of sulphur to the charge in the form of iron pyrite.

For fuel we employ oil burned in an external firebox, which is air-jacketed to prevent loss by radiation. The fire gases which, mixed with the required volume of air admitted through openings in the firebox, at the entrance into the furnace, are kept at 800°F., are sent into the 10-deck roaster on the ninth floor. We do not employ muffles. The fuel consumption in the 25-ton roaster was 5 gal. of oil per ton of tailings treated, frequently less, and we are confident that we can decrease it to 3 or 3½ gal. at the most in the large roaster on account of heat-saving

devices incorporated in its construction. Also, there will be another source of heat provided by the exothermic reactions that take place during the roasting process in the furnace. The additional heat derived from these reactions is quite considerable and has made itself felt to an appreciable degree in the small pilot furnace.

In the new plant, the separation of the copper liquor from the leached calcine will be effected by counter-current decantation in three Dorr thickeners. Directly from the roaster the calcine is discharged into a circular mixing tank where it receives return liquor from one of the thickeners. From the mixing tank the pulp and liquor pass into three Dorr classifiers placed in series to effect a separation between the sand and slime. The slime in the overflow from these classifiers passes successively through the three thickeners, the first of which delivers its clear overflow to the precipitation boxes. As the pulp progresses through the battery of thickeners it is washed by water and waste liquor from the precipitating launders traveling in the opposite direction.

During our test runs with the 25-ton roaster, the extractions ranged from 65 per cent. to 77 per cent. of the total copper in the tailing. For the present the copper in the liquors is precipitated on scrap iron in a system of launders, although electrolytic separation is taken into consideration by the management for the future. Concerning the electrolytic deposition of the copper and a decision to introduce it in our plant, we are looking forward to the results which will be obtained by this method at Ajo in the New Cornelia Copper Co.'s plant now under construction, where the process has been worked out by such pioneers as Dr. Morse and Mr. Tobelmann, who have presented their paper on this subject before this meeting.

B. B. Gottsberger, Miami, Ariz.—For some time we have been experimenting on the recovery of oxidized copper. While trying to convert the oxidized copper to sulphide by means of calcium sulphide and sulphuric acid, we concluded that what conversion was obtained resulted from the solution of the copper in acid and its subsequent precipitation as sulphide. Laboratory work demonstrated that a process along these lines could probably be made to work, but desiring to avoid the use of hydrogen sulphide, if possible, we concluded to look for another method before trying this scheme on a working scale. Early in the present year we did some laboratory work along the lines of the process we have just seen at Chino and at the present time are trying out this idea in an experimental plant handling about 50 tons per day. I am very hopeful of a successful result from this work.

H. E. Williams, Calumet, Mich.—The Calumet & Hecla Mining Co. has been experimenting for some time, under Mr. Benedict's direction, upon the problem of reclaiming copper from the old tailings banks in

Torch Lake, which contain about 40,000,000 tons of recoverable sands. After many laboratory and test-plant experiments, they have finally adopted the ammonia leaching process, and have built a 2,000-ton plant, which is now in partial operation.

The plant has eight unlined steel leaching tanks 54 ft. in diameter by 12 ft. high, provided with the usual wood grating, cocoa matting and canvas filters, removable steel covers and one central and six circumferential discharge gates. After being reground and treated on tables, the tailings are pumped into the plant and deposited in the tanks by revolving distributors. The leaching solution contains about 1.5 per cent. of NH_3 and an equal percentage of CO_2, and after dissolving the copper, is pumped to the still house, where it is treated in ammonia distillation apparatus with low-pressure steam which precipitates the copper as oxide. This is sent to the smelter, while the NH_3 and CO_2 distilled off and condensed is pumped back to the leaching plant to be used again. From sand containing about 10½ lb. of copper to the ton, we recover about 8 lb. We have had to develop a roughing still, which has delayed the still-house work, otherwise the plant would now be operating at its rated capacity. I am sorry I cannot give you a description of the chemical reactions; but I am not a chemist and must leave that for Mr. Benedict in the future.

G. D. Van Arsdale, New York, N. Y.—I would like to ask Mr. Mathewson the reason he adopted wood tanks instead of concrete tanks.

E. P. Mathewson.—We had experimented with wooden tanks, and when we put up the 1,200-ton tanks, we thought the wooden tanks would be satisfactory. After they had been in service for about a year we noticed deterioration, and we had them lined with lead. If they had been built of concrete, they would have been more expensive. For a plant the size of ours, we think the wooden tanks, lined with lead, are preferable; but with larger ones, we think it would be preferable to build them of concrete.

Gold and Silver Deposits in North and South America*

BY WALDEMAR LINDGREN, BOSTON, MASS.

(Arizona Meeting, September, 1916)

I. INTRODUCTION

At the time of the discovery of America the old world had a scant supply of the precious metals. Both the northern and the southern part of the new continent proved wonderfully rich in gold and silver and its treasures were eagerly looted; though the looting has lasted four centuries, the mines of its mountain chains are far from being exhausted. Even the later discoveries in Australasia and eastern Siberia could not rob the Western Hemisphere of its position as the greatest gold and silver producing region of the world, though finally the developments in a narrow and circumscribed area in South Africa wrested from the Americas their supremacy in the production of gold.

Nevertheless the history of the two parts of the great western continent has been strikingly different. At first the Spaniards extracted vast treasures of silver from Mexico, Peru, and Bolivia, while Colombia and some placer deposits in Peru yielded a smaller quantity of gold. A couple of centuries later, a stream of gold began to flow from Brazil, the silver production from the countries mentioned above continuing strong in the meanwhile. Later on, the yield of South America diminished, but to offset this there began a wonderful series of discoveries in North America. The gold fields of California astonished the world; and when the cream of these had been skimmed off there began a no less amazing development of the Central Cordilleran gold and silver districts, which soon made the United States the greatest producer of the precious metals. Aided by ever improving technique, extensive exploration, and a system of railroads, the yield was maintained and increased. Still later followed the discoveries of the gold fields of the arctic region and silenced those who had maintained that the zenith in gold production had passed. Recently the province of Ontario in eastern Canada

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rose unexpectedly with offerings of the richest silver ores the world has known, and with new and at first doubtfully accepted gold fields.

Chile and Bolivia in the middle of the last century added some rich silver mines to their long list of mining districts, and later placer gold began to be extracted in large quantities from the Guianas, but on the whole no such sensational finds were made in the southern continent as had marked the recent history of the northern part, and in many regions the mining of the precious metals fell into a rut, the

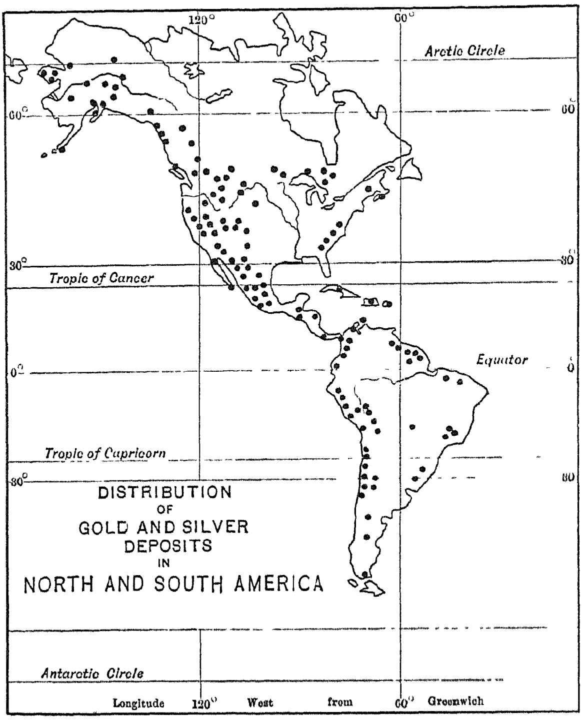

FIG. 1.

production being barely maintained or diminished slowly. The latest events indicate an awakening, and a stimulus under the influence of which the production of South America is gradually increasing. Large amounts of silver are extracted in the copper from operations on a large scale, and dredges dig up the gold of Colombia and Tierra del Fuego.

It cannot be doubted that the total yield of the northern continent of gold and silver is larger than that of the southern part. A glance at the table in the Appendix will show that this difference is strongly emphasized at the present time. During the last decade the gold production of North America had a value of $1,338,268,000,

while South America yielded only $125,000,000. For silver, South America statistics are in less satisfactory shape, but the compilation shows that while in 1913 North America produced this metal to the value of $99,476,400, South America's mines yielded less than one-tenth of this huge sum. Fig. 1 shows the approximate distribution of the gold and silver deposits of the two continents.

There are no better prospectors in the world than those of some South American countries, and we may rest assured that a great percentage of possible discoveries has already been made. Yet no one who has studied South American mining districts can fail to see the possibility of a more extended production than at present, even while realizing the difficulties of climate, altitude, transportation and lack of adequate available capital.

The purpose of this paper is to call attention to the geological features that govern the distribution and richness of the precious metal deposits of South America, to compare them with those of North America, and to classify them according to geological affiliations.

II. GEOLOGICAL FEATURES

A slight acquaintance with the geographic features of the two parts of the American continent suffices for the realization of their essential similarity. The two landmasses, elongated from north to south, have a wide eastern part occupied by fertile plains, hilly country or low mountain ranges, and a narrower western part, with the rough topography of an almost continuous high mountain chain closely following the Pacific Coast, narrow in South America, broadening in North America. This is one of the great earth features, and is known as the American Cordillera. In South America it is also known as the Andes. Considered on a large scale, its build is simple, though in detail it is diversified by two or more parallel ranges, by intermontane high plateaus or valleys, and by volcanoes, many of which are active.

To the geologist this difference of east and west is sharply accentuated, for he knows that the Atlantic side represents the area of quiet where strong mountain-building forces have rested for millions of years—since the close of the Paleozoic era—while the leveling agencies of erosion and sedimentation have been at work. He knows that the western margin marks the long strip of weakened earth crust along which tangential stresses have played since early Mesozoic times. These stresses culminated in the early Tertiary times causing folding and violent thrust faulting, as if an irresistible force had forced a wrinkle in the earth's crust eastward. These Cordilleran disturbances reach their maximum along the inner eastern edge of the chain. To some degree they still continue, accompanied by uplifts and depressions. Lava flows have been poured out in great volume from volcanoes along the Cordilleras, especially on

the western side, and this has continued at least from the early Mesozoic to the present time. At the same time masses of molten rock have been forced up from great depths into the rocks nearer the surface, and cooled there to granites and diorite porphyries without ever reaching the surface, though through gradual wearing away of the covering rocks many such masses are now exposed at the surface.

Almost all primary gold and silver deposits have been formed during or shortly after epochs of volcanic or intrusive activity. Secondary deposits are derived by the disintegration and concentration by water of such primary deposits. They are called placers or alluvial deposits and are usually cheaply and easily worked.

On the American continents the primary gold and silver deposits date from two widely separated ages. The first period is geologically very ancient and belongs to the pre-Cambrian or early Paleozoic; its deposits are thinly scattered over the entire continental area, but are at many places covered by later rocks. The second period is much more recent, and belongs to the late Mesozoic and the Tertiary. Its numberless deposits were formed during the great igneous activity which accompanied the building of the Cordilleras and are thus confined to the western or Cordilleran part of the continents in which area the deposits of the older period are rare because capped by later sediments or flows.

Placers may be formed from deposits of either period.

1. Deposits of the Early Period

Naturally, the deposits of the early period are best observed in the great eastern expanse of the continents where the early rocks are often splendidly exposed. Gold is the principal metal and is always accompanied by quartz gangue. The deposits bear evidence of having been formed at considerable depth and high temperatures. While the majority of these occurrences are poor, yet great richness may be found in small areas, and the purity and coarseness of the gold is favorable to the formation of placers, especially in temperate or warm climates. Wherever continental ice sheets have covered a region, as in Canada, they have almost invariably ground up and scattered the placers.

North America

In North America deposits of this kind are formed in the southern Appalachian States, in South Dakota, in Quebec, in Nova Scotia, and in Ontario. In the latter province the recently discovered Porcupine district presents a case of extraordinary richness, the annual production being now over $4,000,000. The celebrated Homestake mine in South Dakota is working on a pre-Cambrian replacement deposit in form of thick lenses of altered schist with free gold. While containing only

about $4 per ton, the ores yield annually over $5,000,000. A number of scattered deposits of this kind are found in the Cordilleran region of the United States, but they contribute only small amounts to the total production. The most important occurrence is the copper deposit of the United Verde mine in Arizona. Its copper bullion yields a considerable amount of gold and silver.

Compared to the deposits of the Cordilleran or younger period in North America, the yield—both total and annual—is small. Out of an annual gold production of about $130,000,000, the sum to be credited to the old group of deposits is at present (1913) not more than $10,000,000.

Very little silver is obtained from the gold deposits, but a small amount comes from the copper deposits of the Lake Superior districts. Until the discovery of the Cobalt district in Ontario, the proportion of silver in the eastern region to the total output was even smaller than that of gold; but the native silver yielded by this district (almost unique in America) has changed this so that the old deposits of the East are now credited with about 800,000 kg. out of an annual production for North America of over 5,000,000 kg. The great output of the Cobalt district emphasizes again how highly the precious metals may be concentrated within small areas.

South America

In South America we find extremely similar geological conditions, but here the older group of deposits yields decidedly more gold than that furnished by the belt of the Andes. On the other hand, the silver production of the older deposits is insignificant. A somewhat more detailed review will perhaps be acceptable. Fig. 2 shows the distribution of the deposits in South America.

Gold deposits of the older type are known from Venezuela, the three Guianas, Brazil, Uruguay and Argentina. Except in the Guianas they do not form continuous belts, but rather a series of scattered occurrences separated by barren ground or by younger transgressing fluviatile or marine deposits. South of the latitude of Buenos Aires the deposits, if existing, are covered by the Tertiary Pampas formation or by lavas of the same age.

The northeastern region extends 650 miles from the Yuruari basin in eastern Venezuela to the Franco-Brazilian border of the Guianas. The occurrences worked are mostly placers, to the formation of which the conditions are very favorable; but quartz veins or mineralized dikes have also been exploited. The best example of the veins is furnished by the great Callao mine in Venezuela, which, during its life of 30 years (1865-1895), is said to have yielded $28,000,000 in coarse gold. Active exploitation of the placers and some veins is going on in the three Guianas at present, the French colony yielding the greatest amount. In 1912 the

production of this belt was about $5,000,000 and for the last decade it has not been less than $4,000,000 in any one year.

The primary veins from which the placers have been derived are contained in pre-Cambrian schists, diorites, diabases, granites and granite porphyries.

The gold belt seems to continue to the southeast beyond the boundaries indicated, for it is reported that gold occurs in the provinces of

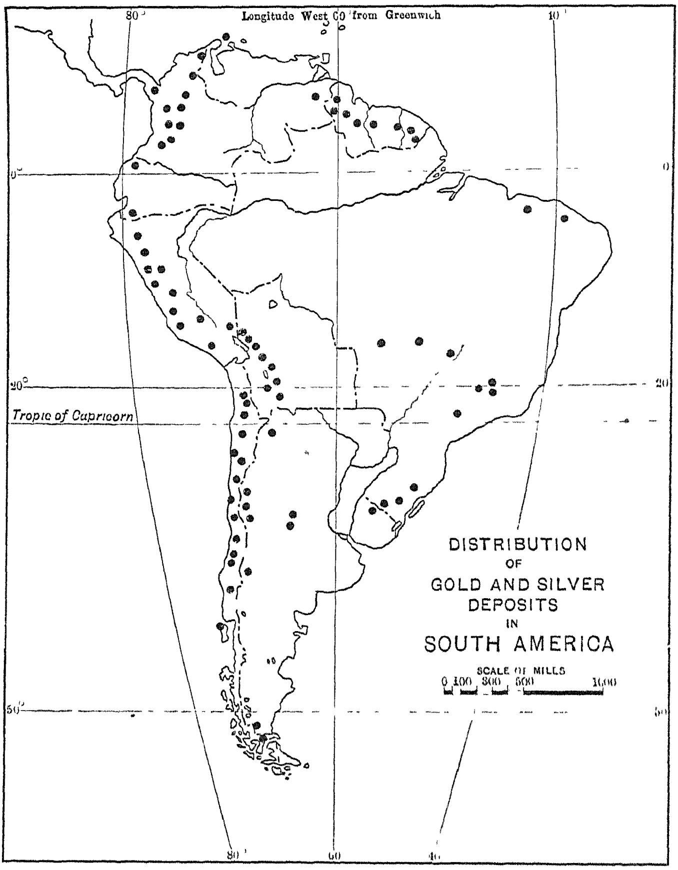

FIG. 2.

Para, Maranhao, and Ceara, in Brazil, beyond the delta of the Amazon River. To the south follows a broad, barren interval until we come to the gold deposits of southern Brazil, in the states of Bahia, Minas Geraes, Sao Paulo, Parana, and Rio Grande do Sul. Of these the state of Minas Geraes is by far the most important. Even in the far western part of Brazil, at Cuyaba in Matto Grosso, occur placers said to be derived from older deposits similar to those of Minas Geraes.

It is almost forgotten at the present time that the placers of southern Brazil yielded heavily in the 18th century, particularly from 1700 to 1775, and this production was particularly welcome at a time when the gold from the Americas seemed exhausted and the treasures of the northern Cordilleras were as yet undreamed of. During the period named, these placers yielded from $1,000,000 to $2,000,000 annually. The total yield during the 18th century is variously given from $200,000,000 to much higher figures. After this period the production languished, but a few quartz mines continued to be operated and a little placer gold was washed. At the present time Brazil maintains its output of gold at from $2,000,000 to $3,800,000 but this is practically derived from three deep mines in Minas Geraes, of which the Morro Velho is the most important, besides having the distinction of being the deepest mine in the world (vertical depth 5,800 ft.).

The deposits are quartz veins of a deep-seated type, allied in places to pegmatite dikes. They occur in part in Archean schists, gneisses and granites, but most of them are found in a thick sedimentary series of schists and quartzite, which is older than the Cambrian but overlies the Archean. This series contains no intrusives, except some pegmatite dikes, and the Brazilian veins are in this respect markedly different from most other pre-Cambrian occurrences. It is believed that igneous intrusions took place in the rocks underlying the pre-Cambrian sediments and that only pegmatitic dikes and quartz veins reached up into the covering series.[1]

Similar geological conditions prevail in Rio Grande do Sul, beyond which the gold-bearing region continues into Uruguay, where the most southerly mines are found near Cuñapirú. Uruguay yields annually up to $100,000 in gold.

The most southerly representatives of this older class of gold deposits appear in the Sierras of the Pampas, for instance, in that extending from San Luis to Cordova in Argentina. The old crystalline schists, granites, and pegmatites here emerge from under the Pampas formation and the Permo-Triassic beds, and contain deposits of tungsten, gold and silver, but the latter two metals do not count in quantities sufficient for economic mining.

While it is possible that some deposits of this kind occur in the pre-Cambrian of the Andean region, which is exposed in Colombia and in the northernmost provinces of Argentina, it is improbable that they contribute perceptibly to the total production.

To sum up: The old gold deposits yield the total production of Venezuela, the Guianas, Brazil, and Uruguay, and at the present time contribute to the gold production of South America about $8,500,000,

[1] E. C. Harder and R. T. Chamberlin: The Geology of Central Minas Geraes, Brazil, *Journal of Geology*, vol. 23, pp. 341 to 378; 385 to 424 (1915).

or not far from the amount extracted from the same class of deposits in North America.

2. Deposits of the Later Periods

General Features

From Cape Horn to Alaska the gold and silver deposits are formed under similar geological conditions, and are of the same general geological age. It has already been emphasized that they are products of the igneous activity which has accompanied the rise of this gigantic mountain chain.

They were formed within several epochs, but all of them lie between the earliest Cretaceous and the present; that is, they are late Mesozoic, Cenozoic or Quaternary in age. They were formed, on the whole, nearer to the surface than the old deposits of the pre-Cambrian, or at least under conditions of more moderate temperature. Many of them, indeed, were formed very close to the present surface. Following intrusions or lava flows, hot waters loaded with gases and metals of igneous origin rose toward the surface, and, in cooler regions of the crust, deposited their load of metals. In part the gold and silver occur in minute quantities associated with copper and lead minerals, and are recovered from the base bullion. Much silver is obtained in this manner, but most of the gold is derived from gold quartz deposits, properly speaking, or from placers caused by the wearing down by erosion of these deposits.

North America

It is difficult indeed to give in a few paragraphs even an approximate idea of the gold and silver deposits of the North American Cordillera. The annual yield of the region is enormous, attaining now $130,000,000 in gold and nearly $100,000,000 in silver.

A great gold producing belt lies along the Pacific and reaches from California to Alaska, with local interruptions. These are the oldest deposits of early Cretaceous age and they have yielded vast placer or secondary deposits. The annual production including the placers, is not less than $40,000,000. Geologically they are connected with the intrusion of dioritic rocks, an intrusion extending like a gigantic dike along the Pacific Coast mountains.

Throughout the interior part of the Cordilleran region are numberless smaller intrusions of granitic or dioritic rocks, or of the porphyries of these rocks, most of them of earliest Tertiary age, some a little earlier, others a little later. Aureoles of gold and silver veins surround these intrusions, and contribute from numerous centers in the interior Cordilleran region to the total production.

Contact-metamorphic deposits, formed where limestone beds have adjoined the igneous contacts and absorbed the emanations from the intrusive magma, add their smaller share to the precious-metal production, but are usually richer in the base metals.

Lastly we have a remarkable type of veins, which occur in lava flows near volcanic vents, and which were formed near the surface by hot springs charged with emanations from the molten rocks. These deposits are often wonderfully rich, both in gold and silver. They are the "bonanza" deposits proper; the Comstock, Tonopah, Goldfield, and Cripple Creek are among the more celebrated localities of such veins; few of them are found north of the Canadian boundary and none of them along the main Canadian or American coast, but they are best represented in Nevada, Arizona, Utah and Colorado. In the United States, they yield not less than $30,000,000 a year.

Going farther south we enter the great mining region of the Mexican plateau. For nearly 400 years an unceasing stream of silver has been poured out of the mines of Mexico, and at the present time the country produces annually about 2,000,000 kg. or 64,000,000 ounces of that metal.[2] Igneous rocks, both flows and intrusions, abound in Mexico, and practically all of the deposits are of latest Cretaceous or of Tertiary age, thus on the whole more recent than many of those of Canada and the United States.

The most celebrated silver mines are of the type formed in or near volcanic flows near the surface: We need cite only Pachuca, Guanajuato, and Zacatecas; but there are hundreds of other similar districts. Of late the annual gold production has risen sharply to $20,000,000 or $25,000,000; part of this comes from silver or base bullion, but the greater part is derived from veins in volcanic rocks similar to those just described and situated at El Oro, in the state of Mexico. It should not be overlooked, however, that there are also in the Cretaceous limestone countless though small intrusive masses of diorite or porphyries around which auriferous or argentiferous veins or contact-metamorphic deposits have formed, and which contribute their share to the production.

The Antilles

Evidences of a feeble mineralization are found in Cuba, Haiti, Porto Rico, and Jamaica and more or less placer gold was obtained, particularly during the 16th Century from the first three islands named. Even now 100 oz. or so are washed annually from the rivers of Porto Rico and

[2] The total production of silver in Mexico is estimated as 122,500 metric tons, a quantity far greater than that yielded by any other country in the world. (See Beyschlag, Krusch and Vogt, *Die Lagerstätten*, vol. 2, p. 69 (Stuttgart, 1912).

perhaps the same amount from those of Haiti. The gold seems to be derived from the vicinity of intrusives such as diorite and serpentine, in part, if not altogether, of post-Cretaceous age. The gold placers of Cuba were situated in the middle part of the island, in the Santa Clara and Puerto Principe provinces. Those of Haiti are said to have been highly productive in the early days of the Spanish régime.

Central America

The Cordillera does not continue as an unbroken chain from Mexico into Colombia. The structure of Central America is complex, with short easterly trending ranges of older rocks in Guatemala and Honduras. Farther south these older rocks are submerged beneath Tertiary and recent lavas, in part andesitic. The isthmus connecting the two Americas is in fact marked by a chain of volcanic cones, many of which are active.

Though some mineralization is found in the older rocks of pre-Tertiary age the valuable deposits are mainly in andesitic or rhyolitic rocks, and belong clearly to the class of veins which were formed near the surface. Some of these yield mainly gold, but in many cases they are of the well-known type in which gold and silver occur together without notable amounts of the baser metals. The annual production of Central America ranges from $1,500,000 to $4,500,000 in gold and from 50,000 to 75,000 kg. of silver.

Guatemala contributes but little though there are many prospects and placers on Motagua River on the Atlantic side.

Honduras has the reputation of great richness. Its placers of Olancho and Choluteca were worked by the early conquerors. At present the greatest part of its production comes from the gold-silver mine of Rosario near Tegucigalpa. The republic is the largest silver producer in Central America. Gold to the value of about $600,000 is produced annually in each of the three states, San Salvador, Nicaragua, and Costa Rica. In Nicaragua rich placers have been worked in the Prinzapolca and other Caribbean rivers, and the gold mining district of Pis-pis in the northeastern part of the republic has lately attracted much attention. Costa Rica has had a considerable production from the placers of Monte Aguacate. The Abengarez and Montezuma lode mines, on the Pacific side, are now the chief producers. In San Salvador the production comes largely from the Butters mines. All of these veins appear to be contained in andesite or rhyolite.

We find the same condition in Panama, though at present there is little production from this state. The Espiritu Santo mine at Cana near the Colombian boundary has been worked from the 17th to the 20th century and the deposit is contained in Tertiary andesite.*

* Malcolm Maclaren: Gold, London, 1908.

The South American Cordillera

General Features.—From Cape Horn to Colombia the South American Cordillera or Andes forms a continuous chain closely following the coast. Its width ranges from 100 miles near Magellan Strait to 500 miles in the latitude of Bolivia. North of Bolivia it again contracts to a width of about 300 miles. It is thus, considering its length, a narrow mountain chain, but nevertheless generally made up of three longitudinal units. In the north they are known as the Eastern, Central, and Western Cordillera or by other local names. In Peru they are spoken of as the Coast, Sierra, and Montaña regions, the last being the eastern slope of the Andes. In the south there are locally four subdivisions: the Coast Range, the Western and the Eastern Cordillera, and the pre-Cordilleras or Front Ranges. Between the eastern and western range lies, in Bolivia, the high plateau or "Altiplanicie." In places, as in northern Chile and in Bolivia, the western range itself partakes of the character of a plateau.[3]

Two ranges stand out by reason of great altitudes, both being rich in mineral deposits. One is the Sierra Blanca of northern Peru, in the Western Cordillera; the other is the Cordillera Real of eastern Bolivia which includes the high summits of Sorata and Illimani. Fig. 2 shows the distribution of the precious-metal deposits in South America.

III. GEOLOGY OF SOUTH AMERICA

Introduction

It will be admitted that it is no easy task to condense in a few pages what is known of the geology of a continent; and for the imperfections and omissions in this account I must therefore ask the indulgence of the reader.

Broadly speaking, the most prominent formations of the Andes are the Cretaceous sediments, which extend almost without interruption from northern Colombia to Tierra del Fuego. Of scarcely less importance, though smaller in area, are the Tertiary and Recent lava flows and the intrusive masses of early Tertiary age. No great intrusions of Cretaceous age appear to exist in South America, although the volcanic activity in the Jurassic and Cretaceous was intense and yielded heavy masses of lava flows intercalated in these formations.

As far as known, the pre-Cambrian is only exposed in the north and on the Argentine side of the Bolivian high plateau.

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[3] Isaiah Bowman: Physiography of the Central Andes, *American Journal of Science*, 4th Ser., vol. 28, pp. 197 and 373 (1899).

Colombia and Ecuador

The work of W. Sievers, A. Hettner, A. Stuebel and Theodore and W. A. Wolf permits a general view of the geology of these countries. As already emphasized, the Andes of Colombia, divided into three chains, do not continue toward the isthmus, but bend eastward toward Venezuela. The coast, both in Colombia and Ecuador, is occupied by Tertiary strata. The Cordilleras consist in general of a core of crystalline schistose rocks which are generally referred to the pre-Cambrian. Above these there is a great break in both countries: The Paleozoic and the early Mesozoic apparently are missing. Instead, the Cretaceous overlies the schists and the extensive beds are divided into the Lower and Upper, the latter being overlain by the Guaderas beds, probably also Cretaceous. There was no marked folding during the Cretaceous. Quartz monzonites and allied rocks are reported from many places; they are older than the Tertiary and younger than the upper Cretaceous. Flows of "Labrador porphyrite" and tuffs are embedded in the Cretaceous.[4]

The Cretaceous is unconformably overlain by the Tertiary. Latites and tuffs represent the volcanic activity of the early Tertiary, continued by the *ejectamenta* of a series of recent volcanoes, most strongly represented in Ecuador.

A sketch map of the general geology of Ecuador, by W. A. Wolf,[5] shows similar conditions. There is a broad belt of Tertiary beds along the coast adjoined by a narrow belt of Cretaceous with associated eruptives. Then follows the volcanic belt, Quito being placed at its eastern margin, and the main Cordillera east of that city is built of granite and crystalline schists, all probably pre-Cambrian.

Peru

Much information on the geology of Peru is contained in the publications of the Cuerpo de Ingenieros de Minas at Lima, which include also some of the important writings of Prof. G. Steinmann. The results refer mainly to the western and central Cordillera, and the geological features of the Montaña slope, clad in tropical vegetation, are as yet little elucidated.

Steinmann's profiles[6] from the Pacific to Rio Marañon show 180 km.

[4] E. Lehmann: Beiträge zur Petrographie des Gebietes am oberen Rio Magdalena, *Tschermak's Mineralogische u. Petrographische Mitteilungen*, vol. 30, pp. 233 to 280 (1911).

[5] Sketch of the Geology of Ecuador, condensed in *Mining and Scientific Press*, vol. 105, No. 4 (July 27, 1912).

[6] Gebirgsbildung und Massengesteine in der Kordillere Südamerikas, *Geologische Rundschau*, vol. 1, Fas. 1–3, 1910.

Ueber gebundene Erzgänge in der Kordillere Südamerikas, International Mining Congress, Düsseldorf, 1910.

of upper and lower Cretaceous beds with interbedded volcanics, strongly folded and in part overturned toward the east. There are in these Cretaceous rocks numerous early Tertiary intrusions of granodiorite and porphyries ("*Andesitische Tiefengesteine*"), but few of them are more than 10 km. in width. In the valley of the Marañon, old ("pre-Devonian") schists and granites appear for the first time and probably form the continuation of the pre-Cambrian of Colombia and Ecuador. The porphyritic intrusions are extremely numerous, and Steinmann refers to them as "laccoliths" though usually they have a vertical attitude, conformable to the surrounding sediments. Farther south the granodioritic batholiths become even more abundant, one exposed in the Rimac River being 50 km. in width. They always metamorphose the surrounding Cretaceous limestone.

Bolivia and Southern Peru

A section across this region, recently described by J. A. Douglas,[7] is 330 km. long but does not include the whole of the Montaña slope. Here the Andes are divided into the Western Cordillera, the Bolivian High Plateau or the "Altiplanicie" and the Eastern Cordillera or the Cordillera Real. The latter includes the highest summits, Illimani and Sorata, but contains no volcanoes or large masses of volcanic rocks. It is largely built of older Paleozoic sediments (Cambrian, Ordovician, Silurian and Devonian), mostly slates and sandstones, and these are intruded by masses of granite, diorite and porphyries. The upper Devonian and the lower Carboniferous are both absent.

In the "Altiplanicie" we find the same folded Paleozoics, with transgrading Cretaceous in part terrigenous sediments, such as those of Coro-Coro. The Cretaceous is covered by post-Miocene andesites.

The Western Cordillera along this section is essentially a volcanic range with numerous dormant or extinct volcanoes, and vast accumulations of lavas including rhyolite, trachyte and andesite.

Underlying these rocks and beautifully exposed along the Chilean coast as far north as Arica, are Jurassic and Cretaceous strata interbedded with contemporary lavas, and intruded by early Tertiary granular rock. The latter range from quartz monzonite to quartz diorites, and are accompanied by pegmatite dikes, many of which carry tourmaline. These intrusives are best exposed in the cañons.

Mr. Douglas regards the intrusive rocks of the Eastern Cordillera as post-Devonian and pre-Jurassic in age; but this is apparently not proved, some authors calling them early Tertiary.

[7] Section across the Andes in Peru and Bolivia, *Quarterly Journal of Science*, vol. 70, pt. I, pp. 1 to 53 (1914).

Chile

Conditions similar to those just described prevail in Chile. We find here, however, a coast-range of lower elevations, largely made up of Mesozoic sediments with interbedded volcanics and a main western range, plateau-like in the north, which is surmounted by a long line of active volcanoes and often really constitutes the western margin of the Altiplanicie. The basement on which the volcanic cones rest is largely of Mesozoic sediments, more or less abundantly intruded by granodioritic rocks. South of Concepcion the intrusive granitic rocks increase in volume and are bordered on the west by metamorphosed sediments of doubtful age in Chiloe island and the Taytao peninsula. Quensel's[8] researches have shown that a vast body of quartz dioritic intrusive, similar to the batholith of British Columbia, but of greater length, follows the coast from Puerto Montt down to the extreme tip of the continent.

On the east side this batholith is almost continuously adjoined by Mesozoic sediments[9] in which great flows of "quartz porphyry," "porphyrites" and their tuffs are embedded. These continue for 1,000 miles or more northward along the eastern slopes. The nomenclature is open to objection; the rocks are rather rhyolites, andesites, etc.

In this Patagonian region the distinction between the coast, central and east Cordillera is less clearly marked. Pre-Cordilleras or front ranges appear on the east side and are made up of granitic laccolithic intrusions. East of these, again, are found vast table-lands of basalt and other Tertiary effusives, which slope eastward and in places reach almost across Patagonia.

In southern Patagonia there is only one period of folding, involving Cretaceous and Tertiary beds, while farther north and indeed through the whole chain of the Andes there are two periods of folding, one Jurassic or older, the other late Mesozoic or early Tertiary.

Argentina

The recent work of Argentine geologists, such as R. Stappenbeck, H. Keidel, and others, has given us a clear idea of conditions along the eastern slope of the Andes. This is rarely a simple slope but usually a succession of ridges, the more easterly of which are called the pre-Cordilleras. In the extreme north, in Salta and Jujuy provinces, really the

[8] *Geologisch-Petrographische Studien in der Patagonischen Cordillera* (Upsala, 1911).

[9] Practically all of the sediments of the region of Magellan Straits and Tierra del Fuego are considered as belonging to the Mesozoic series. On the west coast the batholithic rocks face the sea.

continuation of the Bolivian Altiplanicie, the Paleozoic rests according to Keidel[10] with marked discordance on phyllites and quartzites of probably pre-Cambrian age.

In the eastern main Cordillera the marine Mesozoic (Jurassic and Cretaceous) rests unconformably on the basement of Paleozoic slates and includes great masses of flows of "quartz porphyries" and "melaphyres," *i.e.*, rhyolites and basalts.

In the pre-Cordillera of San Juan and Mendoza[11] there are heavy continental deposits of upper Carboniferous to upper Triassic age, resting on a Paleozoic folded basement. According to I. Bowman and other geologists these "pre-Cordilleras" continue northward into Bolivia and here also consist, in large part, of continental sandstone deposits. Small areas of porphyries and granite are intruded in these rocks. The series is gently folded toward the east.

On the eastern slopes of the Andes, sedimentary rocks generally predominate. Two periods of folding are recognized: an older Paleozoic and a younger Tertiary movement, the latter being designated as the properly Andene disturbances. Along the eastern border the latter is marked by overthrusts and overturned folds.

IV. DISTRIBUTION OF SOUTH AMERICAN DEPOSITS OF GOLD AND SILVER

In the following paragraphs a brief summary is given of the distribution of the precious metal deposits in each of the cordilleran states of South America.

Colombia

In Colombia we find the principal gold belt of the Andes, which under adverse circumstances yields annually a notable production of \$3,000,000 to \$4,000,000. This production is probably capable of considerable expansion.[12] The total yield of that country, as calculated by Vincente Restrepo, amounts to about \$700,000,000; therefore Colombia takes its place among the great gold producing regions of the world.

The deposits are mainly in the western and central ranges, which do not continue northward into Panama, but bend eastward toward Vene-

[10] Ueber den Ban der Argentinischen Anden. Sitzungsberichte der Kaiserlichen-Königlichen Akademie der Wissenschaften (Wien, 1907).

Die neueren Ergebnisse der Staatlichen geologischen Untersuchungen in Argentinien. *Compte Rendu*, 11th Session Congrès Géologique International, Stockholm, pp. 1127 to 1141 (1910).

[11] R. Stappenbeck: La Pre-Cordillera de San Juan y Mendoza, *Anales*, Ministerio de Agricultura, Seççion géologica, Tomo 4, No. 3 (Buenos Aires).

[12] The latest statistics for 1914 show a very marked increase in the production of Colombia, the figure being \$4,678,600.

zuela. The eastern range, in which the city of Bogota is situated, appears to be lacking in precious-metal deposits.

Heavy gravel deposits containing gold and platinum are found along the coast on the Atrato and San Juan rivers, but the richest placers, some of which are now being dredged successfully, lie along the drainage trending northward, in the Magdalena, Porce, Cauca, and Nechi rivers. These are deposits of great value though difficulties of transportation and climate have interfered with their successful exploitation.

The majority of the lode mines are in the departments of Antioquia, Cauca, Bolivar, Tolima, and Santander, of which the first two are the most important.

The deposits are mostly typical quartz veins, often with crystallized native gold, and more or less pyrite, pyrrhotite, arsenopyrite, chalcopyrite, galena and blende, occasionally also tellurides. They are closely related to the California type and undoubtedly allied in their genesis to intrusive rocks. Though the deposits usually occur in granite and schists of probable pre-Cambrian age, porphyries or monzonites of much later date (probably early Tertiary) are usually found close to them. These intrusive rocks have sometimes been described as andesites or rhyolites.[13]

Among the deposits there is also another class, the representatives of which yield gold and silver or silver alone, and which occur in undoubted flow rocks, such as andesite and rhyolite. Many of them contain stibnite, tetrahedrite, pyrargyrite, jamesonite and stephanite and are formed under materially different conditions and near the surface. Such mines are those at Marmato and Echandia in Cauca, and those near Manizales on the boundary of Tolima and Antioquia.

Altogether Colombia must be considered as the most promising gold-bearing region of South America.

Ecuador

Apparently Ecuador is not rich in deposits of precious metals. The coast is occupied by Cretaceous and Tertiary sediments, the former including some intrusive rocks. These are adjoined by a zone of igneous flow rocks of Tertiary or recent age, surmounted by volcanic cones, while, according to W. A. Wolf, the best authority on the subject, the main or Eastern Cordillera is built of ancient schists and crystalline rocks.

Almost the whole of the moderate production of a few hundred thousand dollars comes from the ancient mines at Zaruma near the Peruvian boundary and 50 miles from the coast. According to J. R. Finlay, these veins are contained in a fine-grained diorite. In the Esmer-

[13] H. W. Nicholas and O. C. Farrington: The Ores of Colombia, *Bulletin No. 33, Field Colombian Museum*, 1896.

aldas near the coast and the Colombian boundary there are placer deposits which have not so far been successfully worked; the eastern ranges are also said to contain placers which may be derived from deposits of pre-Cambrian age.

Peru

The conditions in Peru are very different from those in Colombia. There are relatively few gold deposits—some veins are being worked, and a certain amount of placer gold is obtained from the Montaña region of southern Peru. The annual production of gold is rarely over $500,000; thanks to the careful work of the Cuerpo de Ingenieros de Minas it is possible to gain an exact idea as to its derivation. Half of the production comes from the copper of Cerro de Pasco. One-sixth is derived from placers and one-fourth from gold-quartz mines proper.

On the other hand, Peru is the leading silver-producing country in South America, the present annual output being about 9,600,000 oz. or 300,000 kg. Of this again more than one-half is derived from the copper mines of Cerro de Pasco, a small amount from lead bullion, and the remainder from silver or gold-silver deposits.

It is well known that Peru has yielded an enormous amount of silver. Professor Vogt has estimated 35,000,000 kg. as the production from 1533 to 1910. Whether this is accurate or not, it is certain that Cerro de Pasco has contributed the greater part of the silver of Peru.

The silver districts are very numerous and generally situated in the Western Cordillera in the departments of Cajamarca, Libertad, Ancachs, Huanuco, Junin (Cerro de Pasco), Lima, Huancavelica, and Arequipa. It would seem that the silver production could be considerably increased.

Geologically there is also a great difference from conditions in Colombia. In Peru and Chile we find along the coast and Central Cordilleras a strong development of Jurassic and particularly Cretaceous sediments, folded and in part overturned toward the east. These Mesozoic sediments contain embedded lava flows of the same age, which, however, do not appear to be of importance as regards mineralization.

According to Prof. G. Steinmann,[14] the great majority of Peruvian deposits are undoubtedly in close genetic connection with numberless small intrusive masses of "andesite," "dacite," or "liparite." These names are confusing for the rocks are really deep-seated dioritic or monzonitic porphyries not at all connected with the "effusives" or flow rocks.

It is thus clear that practically all of the Peruvian deposits are of the intermediate type, formed far below the surface. It is doubtful whether

[14] Gebirgsbildung und Massengesteine in der Kordillere Südamerikas, *Geologische Rundschau,* Bd. 1, Heft 1–3 (1910).

there are in Peru any deposits of the type of the Tonopah, Comstock, or Pachuca veins.

The great Cerro de Pasco deposits, for instance, occur in or close to a stock of "dacite" or "biotite andesite," which has metamorphosed the surrounding Cretaceous sediments. The proper name would seem to be biotite-diorite porphyry. In their upper levels the veins carried probably secondary silver ores of wonderful richness, while in depth they have been found to contain low-grade copper ores, which now form the basis of a great industrial enterprise.

Besides the smaller bodies of intrusive porphyries, there are also numerous large intrusive masses or "batholiths" of granodioritic rocks. Some of these form the central parts of the great ranges, and they may continue for a long distance with a width sometimes reaching 50 miles. Around these also there has been more or less mineralization, but of a more feeble character than attended the intrusion of the porphyries. The time of intrusion is taken to be early Tertiary.

In the gold-bearing region of southeastern Peru (northeast and north of Lake Titicaca), we find different conditions. Here the folded sedimentary rocks are of early Paleozoic age and more or less intruded by porphyries and granodiorites. This is in the regions of Carabaya and Sandia, and the Inambari basin on the Montaña slope. A very widespread, though not intense, mineralization has taken place; the primary gold deposits are apparently poor but the placers are widely distributed and numerous; partly successful attempts have been made to mine them. This belt is, in fact, the northern continuation of the great tin-silver-gold belt of the eastern range of Bolivia.

Bolivia

Bolivia produces little gold at the present time, but its placers on the Montaña side have at times yielded heavily. They lie on the eastern slope of the great range, east of Lake Titicaca, which counts among its peaks Sorata and Illimani, each over 21,000 ft. in elevation. Celebrated among these were the placers of Tipuani on the east slopes of Sorata, which have yielded great amounts of gold since the time of the conquerors. There are many other localities south of this. Other placers have been worked recently on the San Juan River near the Argentine boundary. At the present time only two gold veins are worked, both in the eastern range and said to be of the "saddle reef" type inclosed in slates and sandstones.[15] The quartz and free gold are accompanied by pyrrhotite, arsenopyrite and pyrite. They thus belong to the intermediate type accompanying intrusive rocks. The ore is of low grade.

[15] F. C. Lincoln: Incaoro Mine, *Mining and Scientific Press*, vol. 108, No. 14, p. 561 (Apr. 4, 1914).

Bolivia points with pride to its production of silver. The yield from 1553 to 1910 is stated to have been 48,800,000 kg., to which the mines of Potosí are said to have contributed no less than 30,000,000 kg making this district the greatest silver mine the world has known. Nor is this large production entirely a matter of the distant past, for it is said that the Compagnie de Huanchaca de Bolivia sent to the markets of the world from its mines, which lie to the south of Potosí, silver and lead to the value of $50,000,000 between the years 1873 and 1888. At present Bolivia yields 80,000 to 150,000 kg. (2,500,000 to 4,800,000 fine oz.) per annum. A large part of this comes as a byproduct from the tin mines; another part is derived from the mines near Huanchaca.

The great mineral belt of Bolivia lies in the extremely rough chain which forms the eastern border of the "Altiplanicie" or high plateaus of that country, a region of Paleozoic folded slates with intrusive cores of diorite, granite and porphyritic intrusions. Volcanoes and lava flows are generally absent. In this range there has been produced a widespread mineralization, in part of gold but more characteristically of the peculiar type of Bolivian tin veins first described by Stelzner, and carrying both silver and tin. All these deposits extending from the Peruvian boundary almost to the Argentina border are certainly of the deep-seated type connected with intrusive rocks. In general, these are porphyritic and may be designated as quartz porphyry or granitic porphyry. In the literature they are frequently referred to as andesite and rhyolite, which usage tends to produce an erroneous impression. There are probably no deposits in Bolivia of the type formed near the surface in flow rocks.

Interesting changes are observed in depth. Just as the Cerro de Pasco silver veins turned into low-grade copper veins in depth, so the wonderfully rich silver veins of Potosí are shown, as the great mountain is penetrated by deep adits, to have been transformed into pyritic tin-bearing veins. The silver production from this district is now of smaller moment than formerly.

Chile

Lack of statistical data makes it difficult to review at a distance the deposits of Chile. The republic of Chile, so progressive in other respects, has made little effort to study or keep account of its mineral deposits.

The narrow strip of coast occupied by the republic is in few places more than 150 miles wide, but extends from the 18th to the 56th degrees of south latitude. From latitude 20° to 36°, a distance of 1,200 miles, this part of the Pacific slope is mineralized in a complex and manifold way, while the remaining distance to Cape Horn contains extremely few gold and silver deposits. This is surely a remarkable feature.

It is not my purpose to describe the great resources in copper which have lately been developed in Chile; these deposits as a rule contain little or nothing of the precious metals. Chile has never yielded very large amounts of gold. At the present time the production appears to be diminishing, as may be seen from Table I, and does not exceed a few hundred thousand dollars per annum. The silver production is a little more valuable, but scarcely reaches 30,000 kg. (960,000 oz.) per annum. At no time has the silver reached the figures of Bolivia and Peru, although the rich deposits of the northern coast during a short period in the 19th century made Chile prominent among silver-producing countries.

The present moribund condition of the industry certainly appears strange when we consider the almost continuous chain of mining districts extending over a distance of 1,200 miles.

The total gold production of Chile from the 16th century up to 1906 is estimated by Herrman [16] at $212,000,000, or less than a third of that of Colombia.

The total production of silver is estimated at 6,600,000 kg., only a small part, it will be observed, of the yield of Peru and Bolivia.

The northern half of Chile shows in general a geological structure similar to that of the Western Cordillera of Peru. The Jurassic and Cretaceous formations are strongly developed with contemporaneous lava flows of great volume. Into these are intruded granite porphyries and diorite porphyries in smaller stocks, as well as many batholithic masses of granodioritic rocks. Both of these kinds of intrusions have brought mineral deposits. There are finally heavy masses of late Tertiary lava flows, and in these we find a few representatives of the type of precious-metal veins which were formed near the surface. The great majority of deposits are associated with intrusive rocks and many of these carry tourmaline with copper and gold, indicating that they were formed under conditions of high temperature. It is necessary to read the descriptions critically, for here, as elsewhere in South American literature, andesite, dacite and rhyolite are names often used for intrusive Tertiary rocks, a survival of the old view that any Tertiary volcanic rock must belong to one of these rock types.

Some gold-bearing veins are found in rhyolite and allied flow rocks, for instance, at Guanaco, southeast of Antofagasta, probably also at Sierra Overa, southeast of Taltal, and at Andacollo, southwest of Coquimbo. Other veins carrying both silver and gold occur, according to Moericke,[17] in andesite flows, in part tuffaceous, for instance, at Batuco

[16] *La Producçion en Chile de los Metales y Minerales, Santiago de Chile* (1903). See also Malcolm Maclaren: *Gold*, p. 662 (London, 1908).

[17] W. Moericke: Einige Beobachtungen ueber Chilenische Erzlagerstätten, *Tshcermaks Min. u. Pet. Mitteilungen*, vol. 12, pp. 186 to 198 (1891).

and Cerro Blanco. According to Moericke all veins of this type seem to have a tendency to play out at a depth of a few hundred feet.

Much more numerous are the gold quartz veins connected with intrusives, such as granites and quartz diorites. We find them at Canutillo,[18] north of Taltal, in diorite intrusive in Cretaceous limestone. Others are found associated with tourmaline and copper ores at Remolinos in Atacama, at Tamaya and La Higuera in Coquimbo, and at Las Condes in Santiago. Another gold belt extends from Coquimbo down to Santiago, and to Rancagua and Talca, south of this city.[19] These quartz veins occur mostly in granite near the contact of schist.

While silver is sometimes associated with gold, the richest silver mines of Chile, which yielded great amounts of the metal in the 19th century, occur as a rule separate in Mesozoic limestone, intruded by or interbedded with greenstones of various kinds. They are characterized by extremely rich ore and antimonial and arsenical silver minerals; some of them also contain silver amalgam. Their genesis is doubtful. The gangue is mainly calcite. In depth these veins also are disappointing and the silver production of Chile is now only a fraction of what it was when these mines were in bonanza.

Among these celebrated districts, mainly situated along the coast, are Huantajaya and Challacollo near Iquique, Chanarcillo (50 miles south of Copiapó), and finally a group of districts including Arqueros and Condoriaco (100 miles south of Copiapó).

The great low-grade copper deposits, such as Braden and Chuquicamata, appear to contain very little of the precious metals.

In remarkable contrast to the northern half, so rich in precious-metal deposits, the southern part of the republic appears to be amazingly poor in mineral deposits. Scarcely any mines are reported from this region except an auriferous vein worked by the Spaniards near Valdivia, and some auriferous beach sands along the coast, for instance, on Chiloe island. Not until we reach the Straits of Magellan are there any producing deposits. At Punta Arenas on these straits and on the eastern side of the Andes there are gold-bearing gravels rich enough to justify dredging. Similar placers are found on the south side of the Straits in Tierra del Fuego. About 1902 a dozen dredges were erected here and for a number of years these gravels have contributed largely to the gold production of Chile, yielding annually up to $100,000. The production has decreased materially during the last few years, owing, it is said, to difficulties in dredging the bouldery deposits.

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[18] S. H. Loram: Notes on the Gold District of Canutillo, Chile, *Trans.*, vol. 35, p. 696 (1906).

[19] E. D. Pope: Gold Mining in Chile, *The Mining Magazine* (*London*), vol. 13, No. 1, pp. 33 to 36 (July, 1915).

The difference in mineralization is intimately connected with a great change in topographical and geological conditions.[20] From latitude 42° down to Cape Horn the Cordillera is invaded by the ocean and by ice. Its westerly margin is cut up into an intricate system of fjords and its summits are clad in the armor of immense ice fields. A huge batholith of granitic and dioritic rocks occupies the whole western range, probably from Puerto Montt to the tip of the continent. This constitutes a striking analogue to the batholith of British Colombia; it is of greater length and its width in many places reaches 100 km. On the east side the ice fields often cover its margins. On the west side the adjoining sedimentary rocks are largely submerged, but on Wellington and Chiloe islands these western sedimentaries begin to appear as metamorphosed schists of uncertain age. All along the eastern side the batholith is intruded in Mesozoic (Cretaceous and Jurassic) rocks. Along the eastern edge of the latter we find again front ranges of granitic laccoliths, such as Cerro Payne, Cerro Balmaceda, etc., most of them consisting of granitic rocks. There is little doubt that the gold placers of Punta Arenas have derived their metal from the mineralization along the eastern side of the great Chilean batholith. It would be strange if this batholith would not be accompanied by mineral deposits. That no such have been found may in part be accounted for by the extensive present and former glaciation which would destroy most placer deposits and to the fact that the region is extremely inhospitable. It would not be surprising if scientific prospecting along the borders of this batholith should lead to the discovery of gold-bearing deposits.

Argentina

The present Argentine production of gold and silver is very small indeed, and the country has never yielded large amounts of these metals.

The Sierras of the Pampas, like that extending from San Luis to Cordova, contain a feeble pre-Cambrian or early Cambrian mineralization, referred to above, but these quartz veins appear to be poor in gold and silver. In the same vicinity there is also evidence of a much later development of gold deposits, perhaps connected with the effusion of Tertiary andesitic lavas, but these veins which have the character of crushed or sheeted zones are also poor in gold.[21]

The whole eastern slope of the Andes from the Bolivian plateau to the latitude of Santiago de Chile shows a relatively feeble mineralization.

[20] P. D. Quensel: *Geologisch-Petrographische Studien in der Patagonischen Cordillera* (Upsala, 1911).

[21] E. Gerth: Constitucion géologica de la Provincia de San Luis, *Anales* Ministerio de Agricultura, Seçion Geologico, Tomo 10, No. 2, 1914.

The slopes of the central Cordillera and the pre-Cordilleras are largely composed of sedimentary rocks folded, overturned, and overthrusted toward the east, with relatively small and inconspicuous areas of igneous rocks,[22] which are designated as andesites and dacites, but which in reality seem to be holocrystalline intrusives. There are also smaller areas of granular rocks of Tertiary age, which were designated as "Anden diorite" by Stelzner.

Gold, silver, and copper prospects are rather abundant, but at very few places has serious work been undertaken. The most important deposit, located at Famatina, is a copper-bearing vein with sulpharsenides and antimonides of copper and very little gold and silver.

The eastern slope of the Andes in the northern half of the Argentine Republic is comparable in a way to the eastern Rocky Mountain chain of Canada. Both show overturned folds and overthrusts toward the east, with comparatively little of intrusive rocks and attendant mineralization.

The gold-silver-tin belt of the Bolivian eastern Cordillera apparently does not enter the Argentine territory.

No lode deposits are reported south of Mendoza, except on the head waters of Neuquen River, at about the latitude of Concepcion in Chile, where there is a mining district of gold-bearing quartz veins in granite of uncertain age. Considerable work has been done on these, but the expected production does not seem to have been realized. The ore is apparently of low grade. The only other precious-metal deposits reported from the eastern slope of the Andes in Patagonia are placers of doubtful value on the headwaters of Chubut, Rio Gallegos and other streams. Placers and some lode mines have been taken up at various places on the Argentine Tierra del Fuego, but little information is available as to their values.

As observed above, the Mesozoic beds of the Patagonian Cordillera and eastern Cordillera are intruded by laccolithic and batholithic masses of granitic rocks, and careful prospecting might well yield favorable results. The glaciation probably would have destroyed any placers which may have existed in this region, and this guide for the prospector is, therefore, generally lacking.

V. COMPARISON OF THE TWO CONTINENTS

It has been shown that the pre-Cambrian and early Paleozoic gold deposits predominate in the eastern part of North and South America; that they are scattered irregularly over a wide territory and do not form well-defined belts except locally; and that the heavy production is very much localized. There is reason to believe that such deposits occur here

[22] R. Stappenbeck: La Pre-Cordillera de San Juan y Mendoza, *Ibid.*, Tomo 4, No. 3.

and there in the pre-Cambrian rocks of the Cordilleran regions, though they are not easily differentiated from the later Cordilleran period of mineralization. We note the marked localization of rich deposits in the Black Hills and in the Porcupine, which may be compared to the strongly auriferous districts of the Guianas and Minas Geraes. We observe, also, that as far as this earliest mineralization is concerned, both continents are about equally rich. No silver deposits of this period, such as are concentrated to such a remarkable degree at Cobalt, Ont., are known from South America.

In the Cordilleran region of South America the principal and almost the only period of mineralization seems to be that of the early Tertiary, while in North America an important series of deposits dates from the early Cretaceous. The batholithic and smaller intrusions in South America all appear to date from early Tertiary, and the evidence of close connection between intrusion and mineralization is cumulatively strong and convincing. The same general principles of association of the two agencies apply in the two continents.

So far, no definite evidence has been adduced that the great lava flows of the Jurassic and Cretaceous contain mineral deposits of that general age. In North America many intrusions—in fact, the greatest batholiths—date from the earliest Cretaceous. No such occurrences are found in South America.

From northern Mexico to Chile the Cretaceous is by far the most prominent of the sedimentary formations, while the Carboniferous limestone, so important for the mineralization of the Cordilleras in the United States, is entirely lacking.

Another interesting feature is the great scarcity in South America of Tertiary deposits of gold and silver occurring in late Tertiary lavas and formed close to the surface. Popularly the majority of deposits in South America are ascribed to this group, and even the latest text-books fall into this error. There are some of these interesting and rich deposits in the southern provinces of Colombia, but none have been recorded in Peru and Bolivia. In Chile they reappear at some places such as Guanaco, Batuco, and Cerro Blanco,[23] but compared to the deposits of other classes they are rare. This is remarkable, when we consider the widespread occurrence in Central America, Mexico, and the Western United States of deposits of the type of Pachuca, Guanajuato, the Comstock, and Tonopah, all marked by certain well-defined characteristics.

A large number of deposits in Colombia, Bolivia, and Chile approach the high-temperature veins by their content of pyrrhotite and tourmaline.

A curious fact is that, so far, no contact-metamorphic deposits are described from Peru and Chile, although metamorphism of the Cretaceous

[23] The copper deposit at the Braden mine near Santiago also appears to belong in this class.

limestone by the granodioritic intrusions is often mentioned. In the Cordillera Real of Bolivia they would hardly be expected, for there the intruded rock is generally a slate or sandstone.

The poverty of the eastern front ranges of the Andes is paralleled by the lack of precious metal deposits in the eastern or Rocky Mountain range of Canada.

North America stands out in its richness of placer deposits derived from veins of the Cretaceous intrusive period. In South America there is no real counterpart to the great placers of California, Idaho, Montana, Alaska, and Yukon Territory.

The placer deposits of the Andes, which were locally rich, were mostly found on the eastern slopes of the eastern ranges, and were derived from gold-bearing veins in Paleozoic slates, with intruded granite porphyries and allied rocks.

Colombia stands out prominently as the most valuable gold-bearing region of the Andes, from which, in spite of many difficulties, we may expect a considerably increased production.

The next region is formed by Peru and northern Chile—a region of very numerous mining districts in which the mineralization is chiefly in the direction of silver and copper with a few gold-bearing localities, which, however, do not seem to be able to achieve great production. No doubt the silver output could be materially increased, particularly where silver occurs with copper. In looking over the numerous gold-bearing districts of central Chile, the student would like to ascertain the conditions which in so favored a country have held back the production to such a marked degree.

The third region is formed by the Cordillera Real of Bolivia with its rich mineralization of tin, silver, and (subordinately) gold. Undoubtedly this region is one of the most promising in South America.

Lastly a striking contrast is presented between the two tips of the Cordilleran chain. At the north is Alaska, rich in gold, at the south is the Patagonian Cordillera, with its gigantic batholith, so promising theoretically, so barren in reality. It is barely possible that theory may be vindicated and that valuable deposits may be found hereafter in this vicinity.

It is difficult to avoid the conclusion that the South American Andes are somewhat less intensely mineralized in precious metals than the corresponding chain in the northern continent, and that even progress and enterprise will be unable to raise its production of gold and silver to approach the figures attained by North America.

APPENDIX

Production of Gold and Silver in the American Continent for 1913

(From the Reports of the Director of the Mint and from tables in *Mineral Industry*)

Value of 1 kg. Gold, $664.60. Silver $19.00.

South America

	Gold		Silver	
	Kilograms	Value	Kilograms	Value
Venezuela	....	$444,800		
British Guiana	2,036	1,353,500		
Dutch Guiana	708	470,400		
French Guiana	4,590	3,050,600		
Brazil	3,392	2,254,700		
Uruguay	120	29,900		
Colombia	4,471	2,971,700	42,100	800,000
Ecuador	612	406,500		
Peru	741	492,300	299,132	5,683,500
Bolivia	40[4]	26,600	81,300	1,544,700
Chile	1,000[4]	664,600	30,178[1]	573,400
Argentina	4[5]	2,600	1,097	20,800
	17,714	$12,168,200	453,807	$8,662,400

Central America

	4,095	$2,721,700	66,427[2]	$1,262,100

North America

Canada	24,976	$16,598,900	990,500	$18,984,000
United States	133,741	88,884,400	2,074,700	40,348,100
Mexico	28,969	19,308,800	2,112,400[3]	40,144,300
	187,686	$124,792,100	5,177,600	$99,476,400
Grand totals	209,495	$139,682,000	5,697,834	$109,360,900

[1] Figures of 1912.
[2] For 1912: See Report of Director of the Mint for Calendar year 1913, p. 247.
[3] Fiscal year 1912–1913.
[4] Estimated; no exact figures available.
[5] Probably too low.

AVERAGE PRODUCTION OF GOLD IN THE AMERICAN CONTINENT FOR THE YEARS 1903–1912.

(From the Reports of the Director of the Mint and from the tables in *Mineral Industry*)

South America		
Venezuela	$245,300	
British Guiana	1,338,820	
Dutch Guiana	578,650	
French Guiana	2,177,540	
Brazil	2,460,150	
Uruguay	66,650	
Colombia	2,884,590	
Ecuador	271,430	
Peru	625,080	
Bolivia and Chile	612,770	
Argentina	107,380	
		$11,368,360
Central America		
	2,535,680	$2,535,680
North America		
Canada	12,179,510	
United States	90,789,060	
Mexico	19,711,050	$122,679,620
Grand Total		$136,583,660

The Emerald Deposits of Muzo, Colombia

BY JOSEPH E. POGUE,* PH. D., EVANSTON, ILL.

(Arizona Meeting, September, 1916)

THE writer visited the Muzo emerald mines in July, 1915, and spent six days in their study. This paper embodies the results of his observations, plus information personally communicated by Robert Scheibe,

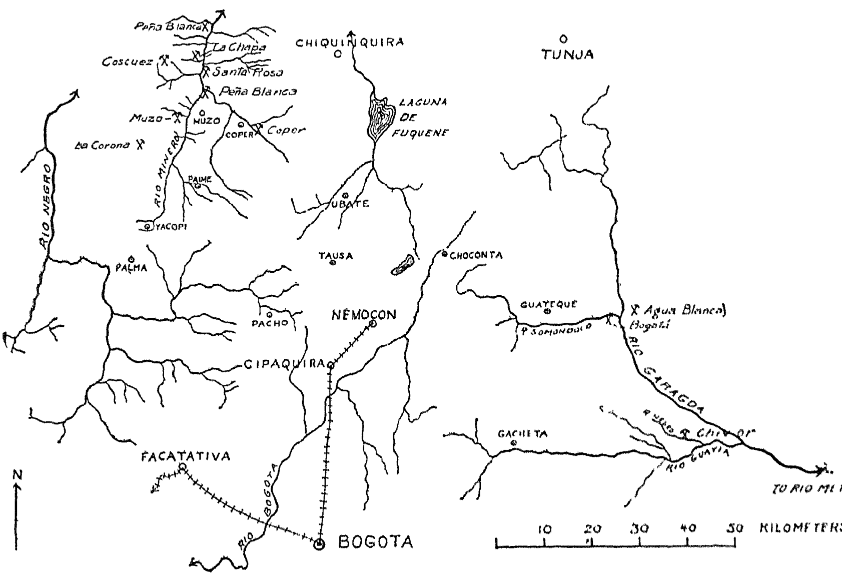

FIG. 1.—SKETCH MAP SHOWING APPROXIMATE LOCATIONS OF ALL IMPORTANT EMERALD MINES AND PROSPECTS IN COLOMBIA.
(Data in part supplied by Robert Scheibe, July, 1915.)

Professor of Geology in the Mining Academy of Berlin, who at the time of the visit was completing a detailed field investigation of nearly a year's duration of the emerald deposits of Colombia. An elaborate account of this valuable work may be expected at a future date from the pen of Professor Scheibe.

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Location

The Muzo emerald deposits are situated in the western foothills of the eastern branch of the Colombian Andes and are distant about 96 km.[1] (60 miles ±) in a direct northwesterly line from Bogotá, the

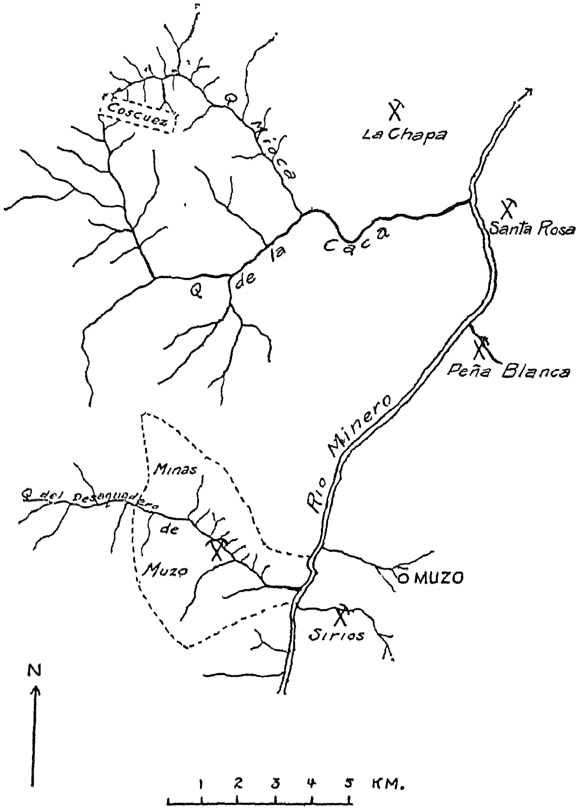

FIG. 2.—SKETCH MAP SHOWING THE MUZO AND ADJACENT EMERALD DEPOSITS. (The Muzo preserve, the property of the Colombian government, is inclosed by broken lines.)

capital of Colombia (Fig. 1). They lie about 8 km. by trail west of the small village of Muzo in the Department of Boyaca, and embrace about

[1] This figure was obtained by scaling off the distance on a map of central Colombia, scale 1/810,000, by William Lidstone, C. E., published by Edward Stanford, London, June, 1899, probably the most reliable map of that region; and checked on the *Mapa de la Republica de Colombia* by Enrique Vidal, Bogotá, 1914, scale 1/2,700,000, the best general map of the Republic. Colombian maps, however, are far from accurate in the modern sense, and on many the Muzo mines (or the nearby village of Muzo) are inaccurately located. In the literature, too, their location is generally erroneously stated, even Bauer (*Precious Stones*, English translation by L. J. Spencer, p. 314, London, 1904) placing them 150 km. from Bogotá, though possibly this figure is intended to represent the distance by trail, a fair estimate in that event.

eight great open cuts, closely grouped, occupying a portion of a steep-walled valley, that of the Itoco, also called Quebrada del Desaguadero, an affluent of Rio Minero which empties into the northward-flowing Magdalena, the great artery of commerce for central Colombia. Though distant but 33 km. from La Dorada, the head of steam navigation on the lower Magdalena, they are inaccessible from that point, and may be reached practicably only from Bogotá via rail to Cipaquira or Nemocon and thence by mule for 2½ days over an execrable trail, nearly impassable in the rainy season[2] (Fig. 1).

The region about the deposits is intensely tropical, characterized by excessive heat and high humidity, with a rank jungle growth that quickly obscures abandoned workings and makes exploration peculiarly difficult and costly.[3] The region round about is sparsely inhabited by Indians who live in squalor and poverty—modified descendants of war-like aborigines, docile and peaceable, even servile, speaking a Spanish *patois*.

The region in general is unhealthful; the natives suffer from tropical anæmia, malaria, dysentery, and other complaints incidental to the latitude. Work in the mines, however, is reasonably safe owing to the excellent location of the workmen's quarters (Fig. 3) and the medical attention and sanitation enjoyed under recent management.

As shown in Fig. 1, the emerald occurs at other points in the mountainous region of Boyaca, but only the Muzo locality has been productive in modern times. The so-called Somondoco deposits (marked Chivor on the map) and those of Coscuez are important historically and enjoy the reputation of being very rich. The other localities indicated are prospects merely, though locally known as *minas*. The total number of emerald localities in Boyaca has been stated to be 157,[4] but this figure is probably a rough approximation. Outside of the Department of Boyaca, the emerald is not definitely known to occur in South America. It has been reported, however, in Colombia near Bolivar, Province of Vēlez, Department of Santander,[5] and tradition points to the Manta Valley near Puerto Viejo in Peru as a source,[6] but it seems probable that all the "Peruvian" emeralds came from the Colombian deposits.

[2] There are two direct routes to the mines: (1) Nemocon to Ubaté to Coper to Muzo; and (2) Cipaquira to Pacho to San Cayetana to Paime to the mines. The first is the better; the mine traffic goes by this route. Outfitting is less inconvenient at Cipaquira than at Nemocon.

[3] The mines lie about 800 m. above sea level.

[4] V. Levine: *Colombia*, p. 116, New York, 1914.

[5] Levine: *Ibid.*, p. 152. This locality is probably authentic, for it is only about 60 km. northeast of Muzo, in the line of strike of the emerald-bearing formation.

[6] Joseph de Acosta: *The Natural and Moral History of the Indies*, reprinted from English translation by E. Grimston, 1604; edited by C. R. Markham, p. 225, London, 1880.

Jorge Juan and Antonio de Ulloa: *A Voyage to South America*, translated from the Spanish, 3d ed., vol. 1, book 6, Chap. 9, p. 466, London, 1772.

FIG. 3.—VIEW OF MINE BUILDINGS AND LARGEST OPEN CUTS AT THE MUZO MINES.

History

The present section is preliminary to a fuller study under preparation on the history and archeology of the emerald in South America.

The early history of the Muzo deposits is buried in the remote past. These, in common with the Somondoco (Chivor) and Coscuez deposits, had long been worked when the Spaniards first set foot in the New World.[7] The three formed the principal, probably the sole, sources of the precious green stone that the Spaniards found so widely distributed through northwest South America and particularly in the realm of the Incas in Peru—a product that shared with gold in the rôle of inciting the cupidity of the invaders and enticing them to brave the dangers of the unknown interior.

The Spaniards acquired their first indication of the source of the emerald in 1537, when Gonzalo Jimenez de Quesada, the conqueror of interior Colombia and the founder (in 1538) of the city of Bogotá, entered the valley of Guachetá in the Chibcha domain and received nine emeralds as a gift from the Indians.[8] He followed up the belief that these were a local product and after the lapse of more than a year succeeded through two of his captains in locating their source in a wild region about 95 km. west of south from Tunja, then the chief city of that region.[9] This locality, now known as the Somondoco[10] (or Chivor), was found to be under exploitation by the Indians, who worked there regularly during the rainy season;[11] but, although every prospecting effort was rewarded, the Spaniards do not seem to have undertaken immediately any sys-

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[7] Vicente Restrepo (*Los Chibchas antes de la conquista española*, vol. 1, p. 125, Bogotá, 1895) as a result of his archeological study of the Chibcha Indians who inhabited the plateau region east of the Muzo deposits, says: "The Chibchas valued emeralds highly, and as the mines of Muzo which produced the finest of these stones were in the hands of their enemies, the Muzo Indians, they exploited the deposits of Somondoco, which were very rich before the Conquest."

[8] H. A. Schumacher: Ueber die columbischen Smaragden, *Zeitschrift der Gesellschaft für Erdkunde zu Berlin*, vol.10, p. 42 (1875).

[9] Detailed, but slightly variant, accounts of the expedition to the mines may be found in Fray Pedro Simon: *Noticias Historiales de las Conquistas de Tierra Firme en las Indias Occidentales*, Bogotá, 1891; and in L. Fernandez de Piedrahita: *Historia General de la Conquista del Nuevo Reyno de Granada*, Antwerp, 1688; Bogotá, 1881.

[10] Some confusion of terms has arisen. These deposits are near the valley of Chivor, and have been called the Chivor deposits. Under Spanish dominion they came to be known as the Somondoco deposits, from the fact that the emeralds mined were sorted in the village of Somondoco; and by this name they are generally called in the literature. They do not, however, occur on Rio Somondoco, but other less important deposits are known on that stream.

[11] Although the statement to follow needs substantiation by further archeological research, the Somondoco deposits seem to have been the most important source of the emerald in pre-Spanish time, overshadowing in output the Muzo mines.

tematic work in these deposits, deterred no doubt by the inhospitality of the region and the difficulties of approach.[12]

Although they soon conquered the Chibcha Indians, the Spaniards met with stubborn resistance from the Muzos, an uncultured and warlike tribe, who like their neighbors, the Chibchas, were in possession of emerald deposits, but were more successful in keeping from their white enemies a knowledge of the localities. They hid all traces of emerald mining for 20 years, though in 1555 Luiz Lanchero founded the village of Muzo in their midst, incited, it seems, by the belief, based on rumor and more pointed information, that the nearby Itoco mountains had yielded the emerald in abundance.[13] In 1558 mining operations were begun by the Spaniards in these mountains and for a time actively prosecuted in the face of repeated attacks by the Indians. The locality, however, was subsequently abandoned and the mine became overgrown with jungle, and its position has not since been discovered.[14]

About the year 1594 the Spaniards succeeded in finding the Indian workings nearby at the site of the present-day Muzo mines,[15] and then

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[12] Schumacher: *op. cit.*, p. 42. Later the Conquerors turned their attention anew to these deposits and carried on extensive mining operations, if we may judge from the reports of W. Lidstone: *Emerald Mines of Chivor*, Colombia, Bogotá, 1906, manuscript, copy on file, U. S. Geological Survey, and E. B. Latham. The Newly Discovered Emerald Mines of "Somondoco," *School of Mines Quarterly*, Columbia University, vol. 32, pp. 210 to 214 (1910–11), who describe extensive tunnels, ditches, and other excavations found at the locality. Lidstone indeed cites the account of a Spaniard (Alonzo Solis de Valenzuela; no reference) as authority for the statement that 1,200 men were employed in the mines, even in 1565. The Somondoco deposits were lost following the War of Independence and were only recently rediscovered.

[13] Schumacher: *op. cit.*, p. 43.

[14] The statements above are based principally upon Schumacher, who seems to have made a fairly thorough search into a number of original sources not yet available to the present writer. Acosta (*Compendio histórico del descubrimiento y colonización de la Nueva Granada*, Paris, 1848; Bogotá, 1901, English translation in manuscript by Isabel Sharpe Shepard, Chap. 18) gives a somewhat different version: He writes that a few years after the founding of Muzo, the Spaniards discovered the present Muzo deposits and began mining operations there in 1568; a rich vein was found prior to that date at Abipí, 2½ leagues distant, but was not worked, for lack of water, and its whereabouts became lost.

Charles Olden (Emeralds: Their Mode of Occurrence and Methods of Mining and Extraction in Colombia, *Transactions of the Institution of Mining and Metallurgy* (1911), vol. 21, p. 194) writes that at the outset operations were confined to the Coscuez deposits, but the present writer has thus far been unable to find any historical basis for this statement. The Coscuez deposits, according to Dimas Atuesta (*Informe relativo levantamiento de los planos y mensura de los Terrenos de las Minas de Esmeraldas de Propriedad de la Nacion*, manuscript, 1898, English translation on file, U. S. Geological Survey) were discovered by the Spaniards after the Muzo mines were under operation; they were then worked for a period, abandoned, reworked preceding the War of Independence, abandoned, and recently rediscovered.

[15] According to Atuesta (*op. cit.*), Captain Juan Penagos was the discoverer.

commenced active work and for some 15 years or more the output was considerable.[16] The production, however, soon fell away, owing to labor difficulties,[17] and toward the middle of the 17th Century the Spanish Crown, whose fifth portion[18] had dwindled, reorganized the industry under the direction of the Royal Treasury. Its administration seems to have been singularly inefficient; excessive labor in the mines was imposed on neighboring tribes, a burden resulting in heavy mortality and serious depopulation of the region; dishonesty on the part of both workers and officials[19] still further lessened the output; and the galleries that were earlier worked were abandoned for open-cut operations, a change not immediately productive of results. Ineffective mining continued to the middle of the 18th Century or thereabouts, when a disastrous fire terminated activities for a time. Work was later resumed but prosecuted only in a desultory fashion until the success of the War of Independence in 1819 transferred the holdings to the new-born Republic. The republican Government at the outset lacked the organization necessary for running the mines, but realizing their possibilities as a source of revenue, soon contracted for their private exploitation, the terms being 10 per cent. of the net profits.[20] The mines were worked under lease from 1824 to 1848,[21] when Congress in Bogotá decreed that all emerald deposits found in the country should be worked under the direction of the nation.[22] This decree does not seem to have been strictly adhered to, for contracts with private parties were subsequently entered into by the Government, some being in the nature of partnerships, others being strict concessions.

It would be scarcely profitable, even if trustworthy data were available, to follow in detail the vicissitudes of the various arrangements made from 1848 to 1909.[23] Suffice to say that the Muzo mines were worked almost continuously during that period, but their development suffered from a lack of any sustained policy of administration as well as from the

[16] By 1612, according to Simon (cited by Acosta, *op. cit.*, Chap. 18), emeralds to the value of 300,000 pesos had been deposited in the royal coffers as the fifth portion accruing to the Spanish Crown.

[17] Fresle (*Conquista y descubrimiento del Nuevo Reino de Granada*, p. 228, Bogotá, 1859), writing of conditions in the year 1636.

[18] According to a royal edict of 1504, a fifth portion of the mining products of the Spanish colonies went to the Crown.

[19] Schumacher: *op. cit.*, p. 45.

[20] Schumacher: *op. cit.*, p. 48.

[21] One of the first lessees was J. J. Paris, the close friend of the powerful Bolivar. In 1828, Paris obtained the concession for himself alone, which he maintained until 1848.

[22] Schumacher: *op. cit.*, p. 49.

[23] M. Ancizar (*Peregrinacion de Alpha por las Provincias del Norte de la Nueva Granada*, en 1850, I 51, p. 50, Bogotá, 1853) visited the Muzo mines in 1850 and found hem under "active and intelligent exploitation." This could not be said of many égimes, until very recent years.

want of engineering and geological advice. In 1909, the Government closed a partnership contract with an English company, The Colombian Emerald Mining Co., Ltd., controlled by South African diamond interests,[24] and the deposits were actively exploited for a time; but after a few years the contract was rescinded[25] and the Government reassumed sole control of the mines. Operations have been totally suspended, however, since Jan. 1, 1913, owing to the fact that the appropriations for administration and exploitation, being entirely insufficient for reëstablishing mining activity,[26] were applied merely to the maintenance of the property. The European War, in its effect on the precious stone market, precludes profitable exploitation in the immediate future.

General Geology

The geology of the Eastern Cordillera of the Colombian Andes is known only in the most general way.[27] The principal geological formations exposed in the emerald-bearing region are shown in the following columnar statement:[28]

Quaternary
Tertiary
Cretaceous

1. Red sandstone with septarian nodules.
2. Compact sandstone; gray fossiliferous limestone between two layers of gray shale with plant impressions.
3. Black, carbonaceous shale and shaley limestone. Carries Muzo emerald deposits and Cipaquira salt deposits.
4. Siliceous schists and conglomerates, with jasper, flint, etc.

These rocks are compressed into great north-south folds and igneous phenomena are largely lacking.

Detailed Geology

General.—The geological relations of the emerald deposits are well exposed in the great open cuts made in exploitation of the emerald veins,

[24] P. J. Eder: *Colombia*, p. 160, London (1913).

[25] As a result, suit was brought in the English courts and judgment was rendered according to which a settlement was reached upon the payment of a consideration on the part of the Government to the emerald company. See Message of the President of Colombia to the Congress of 1914, July 20.

[26] Message of the President of Colombia to the Congress of 1914, July 20.

[27] No detailed studies have thus far been published. The best general accounts are those by A. Hettner (Die Kordillere von Bogotá, *Petermann's Mitteilungen, Erganzungsheft* 104, 1891) which includes a generalized geologic map; and Hans Stille: *Geologische Studien im Gebiete des Rio Magdalena*, with geologic cross-section, Stuttgart, 1907.

[28] In part from personal communication of Dr. Ricardo Lleras Codazzi.

but the surrounding conditions are almost completely hidden by dense jungle growth (Fig. 3). The emeralds are found almost entirely in

FIG. 4.—VIEW OF EMERALD FORMATION IN OPEN CUT, SHOWING CONTORTED CHARACTER OF FORMATION, POSITION, AND RELATIVE ABUNDANCE OF WHITE, EMERALD-BEARING CALCITE VEINS.
(Face about 50 by 75 m.)

calcite veins that traverse a black, carbonaceous, rather intensely folded formation consisting of thin-bedded shale and limestone (Fig. 4).

This emerald formation[29] lies discordantly[30] upon steeply dipping strata, barren of emeralds, composed of heavier beds of carbonaceous limestone intercalated with black shale, and called the *Cambiado* from the Spanish word *cambiar*, to change. Between the emerald formation and the *Cambiado* and ever in close proximity to the plane of discordance are three rock types of great significance in furnishing direct evidence of the origin of the emeralds. These are (1) albite rock, (2) a light-gray rock composed of a soft granular aggregate of calcite, dolomite, quartz, pyrite, and other minerals, called by the miners *Cenicero* (Spanish *cenicero* = ash) in allusion to its ash-like appearance, and (3) aggregates of large, well-formed calcite rhombs in a fine-grained matrix, forming rock masses known locally as *Cama*, from the Spanish word *cama*, meaning bed.[31] In addition, a few pegmatite veins have recently been discovered in the deposits.

The Emerald Formation.—The emerald formation, Figs. 5, 6, 7, consists of thin beds (averaging 2 cm. in thickness) of shale and limestone

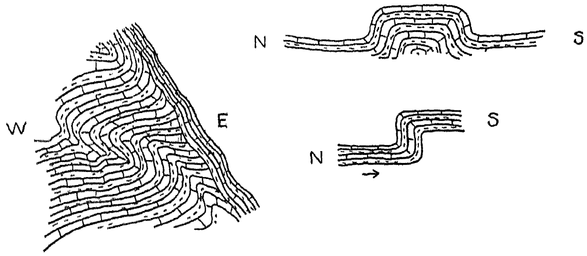

FIG. 5. SKETCH SHOWING CHARACTER OF FOLDS NOTED IN EMERALD FORMATION IN BANCO LIMON.

alternating, the shale in predominance. The shale is a dense, black rock, soiling the hands with excess carbonaceous matter, and most of it effervesces with acid from the presence of calcium carbonate. The limestone is likewise black with carbon but differs from the shale in carrying calcium carbonate in excess of silicate material.[32]

The shale-limestone beds are gently to severely folded, in places contorted (Figs. 5 and 7). The folds are small, irregular in strike and

[29] Known locally as *capas esmeraldiferas*.

[30] This term is used in lieu of "unconformably," because the latter implies an erosion interval; the "plane of discordance" under consideration is interpreted as due to overthrust and it is not definitely known whether younger beds were thrust over on to older, or older on to younger, or part of a formation on to another part of itself.

[31] It seems advisable to retain the Spanish terms in the following descriptions.

[32] Much of the emerald formation might be called with equal propriety either calcareous shale or argillaceous limestone. The silicate material present is shown by the microscope to be, in part, finely comminuted, unweathered, colorless, mineral fragments, probably quartz and feldspar. Possibly some of this was introduced by the mineralizing solutions.

)itch, non-persistent, and lie in all directions; in short, they indicate no ;eneral direction of compression. Their disposition suggests local rather

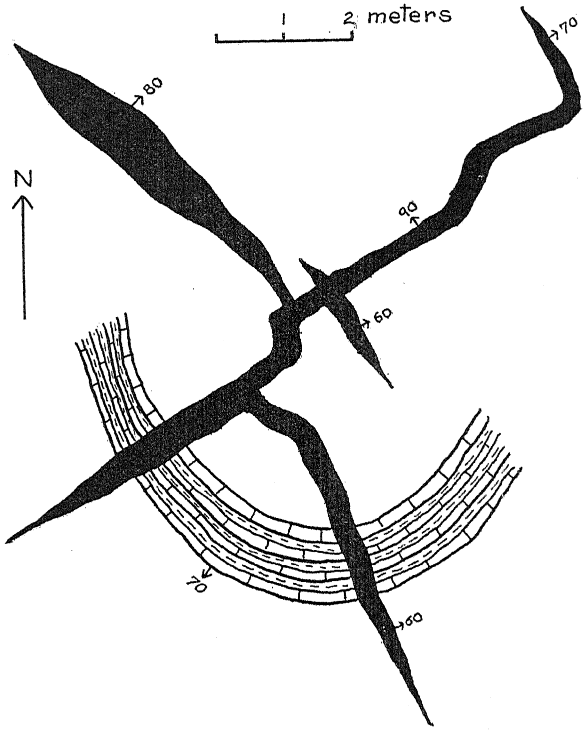

FIG. 6.—SKETCH (IN PLAN) SHOWING TYPICAL RELATION OF EMERALD-BEARING CALCITE VEINS TO EMERALD FORMATION IN "LOS CHULOS" OPEN CUT.

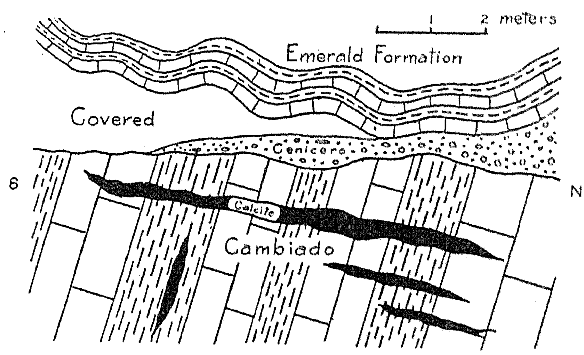

FIG. 7.—GENERALIZED SKETCH (IN SECTION) SHOWING RELATIONS OF EMERALD FORMATION, "CENICERO" AND "CAMBIADO."

than regional pressure. Fractures are prominent, for the most part healed by calcite and consequently marked by veins and seams of white.

Well-defined joints are inconspicuous; faults are present, but, with the exception of the plane of overthrust separating the emerald formation and the *Cambiado*, are for the most part not easily traced.

The emerald is seldom found in the shale or limestone alone. Its usual home is the calcite veins. In many places the beds carry nodules of pyrite or seams of that mineral in well-crystallized condition; and some phases are shot through with well-formed pyrite crystals.

The emerald formation rests discordantly upon the *Cambiado*.

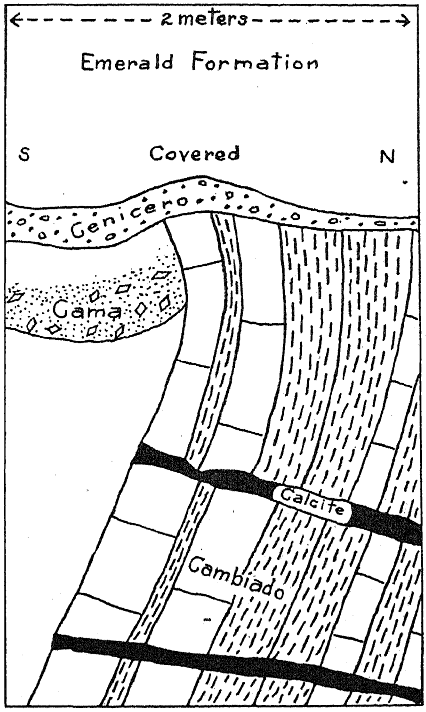

FIG. 8*a*.— FIG. 8*b*.—
PHOTOGRAPH AND KEY-SKETCH SHOWING THE "CENICERO," "CAMA," AND "CAMBIADO," AS EXPOSED BETWEEN BANCO CENTRAL AND LOS CHULOS.
(Note drag and calcite vein in fault plane in the *Cambiado*, indicating overthrust from north to south followed by mineralization.)

The plane of discordance is usually found to be sharp and clear-cut; in many places it is marked by the presence of albite rock, *Cama*, or *Cenicero*.

The Cambiado.—(See Figs. 8*a* and 8*b*). This formation consists of beds of black, crystalline limestone, averaging in thickness about 25 cm. and alternating with thin-bedded shale similar to that of the overlying emerald formation. The limestone shows itself under the microscope to be composed of ragged, granular masses of calcite, inclosed in black carbonaceous matter, and carrying a few to many fragmental crystals

of albite. This rock in places grades upward into a phase in which albite predominates, the so-called albite rock described later; downward it grades presumably into albite-free limestone, but only the topmost few meters of the *Cambiado* are in any place exposed.

The beds dip steeply, in general about 60°, to the south, and are fairly uniform in structure wherever exposed. They are traversed by calcite veins, some following fault planes as in Fig. 8*a*, barren of emeralds, and carry seams, nodules, and scattered crystals of pyrite but in less quantity than the emerald formation. Some thin albite veins have been found[33] in the *Cambiado* of Banco Central. The discordance between the uniform dip of the *Cambiado* and the contorted character of the emerald-bearing strata is conspicuous in many exposures; in some the line of separation is difficult to detect.

The surface of the *Cambiado*, that is, the plane of discordance, is exceedingly irregular, characterized by sudden changes in inclination,

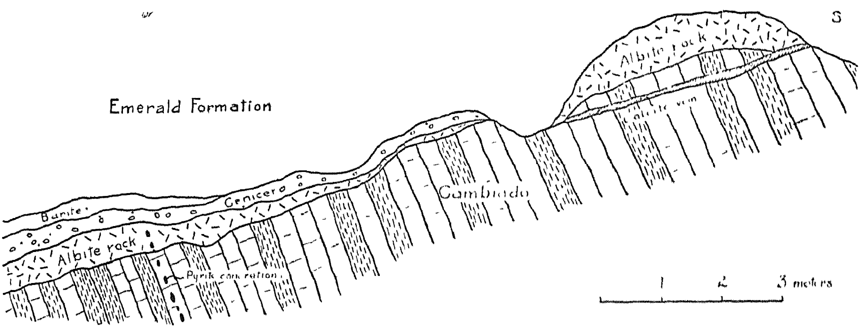

FIG. 9.—SKETCH SHOWING THE RELATIONS OF THE ALBITE ROCK TO THE "CAMBIADO" AND THE "CENICERO" (NORMAL AND BARITIC). EXPOSURE NEAR FOOT OF BANCO CENTRAL.

steep slopes, and prominent hollows. Its general shape is that of a huge bowl-like basin with undulating bottom. Its topographic configuration is so complex, its horizontal development so subordinate, that, apart from the fact that the overlying beds are intensely folded while the underlying ones have uniform inclination, it seems evident that the contorted emerald-bearing strata have been shoved bodily into their present position upon the *Cambiado*. The presence of some brecciated shale and marked mineralization along this plane, as well as evidences of faulting and drag in the upper part of the *Cambiado* (Figs. 8*a* and 8*b*), corroborates the interpretation of overthrust.

Albite Rock.—In places the upper edges of the *Cambiado* strata grade into a granular, grayish rock in which laths of plagioclase feldspar from 1 to 3 mm. in length are thickly set in a greenish-black ground. The microscope shows the feldspar as clear laths and irregular crystals of

[33] By Scheibe.

albite dominating a groundmass of calcite dusted and stained with carbonaceous material; the albite is distinctly later than, and replaces, the calcite. Some portions of the rock are almost wholly albitized; other portions, in addition to albite, carry disseminated rhombs of yellowish

FIG. 10.—PHOTOGRAPH SHOWING THE EMERALD FORMATION (HERE IN HORIZONTAL BEDS) UNDERLAIN BY THE "CENICERO" (GRAYISH-WHITE) AND THE "CAMA" (NOTE CALCITE RHOMBS).

(Exposure between Banco Central and Los Chulos. Height of picture, 5 m.)

dolomite; in places albite seams 5 mm. in thickness are found; and small druses carrying splendid crystals of albite as well as crystallized calcite and dolomite are not uncommon.

The albite rock clearly represents a phase of the *Cambiado* limestone

that has been metamorphosed by mineralizing solutions (or vapors), part of the calcite having been displaced and albite added.

The relation of the albite rock to the *Cambiado* and also to the *Cenicero* is shown in Fig. 9.

The Cama.—This formation is composed of conspicuous rhombs and rhombic twins of calcite, most of them from 5 to 10 cm. in diameter, set in calcareous cement along with some quartz, the whole forming a breccia-like mass (Fig. 10). The habit of the calcite, which occurs as unit rhombohedrons alone or modified by base, and as twins of the first-named form with $(10\bar{1}0)$ as the twinning plane, probably reflects the temperature range of development.[34] This formation rests directly upon the *Cambiado*, but is not continuous, being found only here and there, either alone or close to the somewhat similarly occurring *Cenicero* (see Fig. 10).

The *Cama* in places shows plainly a connection with calcite veins both in the emerald formation above and in the *Cambiado* below. Some calcite veins in the latter have the peculiar calcite crystallization of the *Cama*.

The Cenicero.—This formation occurs as irregular lenses or beds up to a meter or so in thickness present in many places between the emerald formation and the *Cambiado*, either with or without the *Cama* (Fig. 10). It is connected below with the albite rock, into which it locally grades, but unlike the *Cama*, shows no connection with the calcite veins traversing the overlying and underlying formations. In a few places it was noted forming vein-like bodies in the emerald formation itself.

The ordinary *Cenicero* is a crumbly, light-gray aggregate of crystals, chiefly of calcite, dolomite, quartz, and pyrite. A typical specimen under the microscope shows the minerals noted as well-formed, fragmental, and rounded crystals, set in a fine-grained ground, difficultly resolvable, but probably mainly calcareous matter, stained with a little carbonaceous matter. Toward its top the *Cenicero* in many places carries abundant fragments of black shale a centimeter long and smaller, forming masses of breccia rarely seen over 2 m. in thickness.

There are three more or less strongly marked phases of the *Cenicero*—dolomitic, pyritic, and baritic—the normal sequence upward being in that order (Fig. 9). In addition, the lowermost part in many places is albitized, while the topmost portion is nearly everywhere connected with the emerald formation by the breccia phase just noted. The baritic phase is locally seen as an almost pure layer of massive to nodular barite, with a maximum thickness of about 40 cm.

Pegmatites.—Pegmatite dikes were discovered in 1915 by Robert Scheibe in the *Cambiado* near Banco Amarillo and in a ravine back of

[34] Crystal habit is often controlled by external conditions attending crystallization and therefore will become significant when we possess criteria for interpreting it.

Banco Central. The last-named locality was visited[35] and the pegmatite, here about 2 m. in width, found to consist of quartz, in part well crystallized (the low-temperature form), and decomposed feldspar, together with a few crystals of albite and apatite, much greenish to clear allophanite, many small form-rich crystals of pyrite, and a little hyalite.

Minerals

The Muzo deposits present a notable assemblage of minerals, many of them well developed crystallographically and some of particular chemical interest. The present section assembles the geologically significant characteristics of these minerals, but attempts no detailed mineralogical description. A good crystallographic study was published in 1904 by H. Hubert[36] and an accurate mineral list with brief characterizations in 1915 by Lleras Codazzi.[37] The statements here given are the results of the writer's observations, except where otherwise noted.

Emerald.—Usually found in pockets, or embedded, in calcite veins traversing the emerald formation; rarely embedded in that formation itself or in the *Cenicero*. Closely associated minerals forming the emerald gangue are: Calcite, dolomite, parisite, pyrite, quartz, barite, fluorite, and apatite, the last three very rare. The emerald occurs as six-sided prisms with base, some with rarer forms also. Few crystals are larger than the thumb. Most crystals are clear when first taken from the matrix, but later develop cracks; some fall to pieces upon removal.[38]

Choice specimens show a rich green color surpassed by the product of no other locality. Some crystals display zones of color; a few are dark to black with inclusions of carbonaceous matter. In some specimens recently found, the carbonaceous matter is arranged in a six-rayed figure centering about a tapering hexagonal core. One such specimen was examined optically in basal section and proved to be of the same orientation throughout; it therefore does not represent a twinned crystal as suggested by Lleras Codazzi.[39] Its re-entrant angles are presumably the effect of solution and the disposition of the carbonaceous inclusions, the expression of crystallizing forces, as shown also, for example, in chiastolite.

Calcite.—Forms emerald-bearing veins in the emerald formation and barren veins in the *Cambiado*, and is well crystallized where occurring in vugs. Crystals are water-clear to opaque from disseminated carbon;

[35] Under the guidance of Prof. Scheibe.

[36] H. Hubert: Sur les Minéraux, associés à l'Emeraude dans le Gisement de Muso (Nouvelle-Grenade), *Bulletin Musée d'Histoire Naturelle*, vol. 10, pp. 202 to 208, Paris.

[37] Ricardo Lleras Codazzi: Los minerales de Muzo, *Contribución al Estudio de los Minerales de Colombia*, Imprenta de la Cruzada, pp. 3 to 7, Bogotá (1915).

[38] Max Bauer: *Precious Stones*, Spencer's translation, p. 315, 1904.

[39] *Op. cit.*, pp. 6 to 7.

they show a rich variety of forms, with two dominant habits, rhombohedral and prismatic. Closely associated with emerald, pyrite, and parisite. Occurs in the *Cama* as conspicuous unit rhombohedrons (some modified by base) and as rhombic twins, twinning-plane ($10\bar{1}0$). Is an important component of the *Cenicero* as small rhombs and grains.

Dolomite.—Occurs as small, transparent, honey-yellow rhombs in the emerald veins and in the *Cenicero.*

Ankerite.—Occurs as yellowish or brownish rhombs in the emerald gangue; one variety has been reported as carrying a little $CeCO_3$.[40]

Pyrite.—Occurs in well-formed crystals in the emerald veins; as seams, disseminated crystals, and concretions in the emerald formation; as crystals in the *Cenicero;* and as crystals and concretions in the *Cambiado.* The crystals, which range in diameter from a fraction of a millimeter to several centimeters, show a profusion of crystal forms and present three habits, cubic, octohedral, and pyritohedral.

Parisite.—This rare mineral, of the composition $(CaF)(CeF)Ce(CO_3)_3$,[41] was first discovered in the Muzo mines by J. J. Paris, a lessee of the deposits, and named in his honor by Bunsen,[42] who investigated the mineral. It occurs as crystalline masses and crystals in immediate association with the emerald. The crystals are double hexagonal pyramids, with or without the base, and most are under 1 cm. in length.

Quartz.—Occurs as well-formed colorless to greenish (rare) crystals in the emerald veins; inclusions of parisite, pyrite, and emerald have been noted.[43] Less perfect crystals are found intergrown with calcite rhombs of the *Cama.* Crystals rich in forms occur in the pegmatite vein back of Banco Central; these are low-temperature quartz, formed below 575°.[44]

Fluorite.[45]—Occurs as small, colorless to greenish crystals in some of the emerald veins; very rare; forms are cubes or cubes modified by small octahedrons; inclusions of emerald noted.[46] Has also been noted by Scheibe (oral communication) in the albite rock at one spot.

Apatite.—A few, well-formed, glassy crystals have been found by Scheibe in the emerald gangue and in the pegmatite vein back of Banco Central.

Gypsum.—Present in the emerald formation as well-formed, clear,

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[40] Lleras Codazzi: *op. cit.*, p. 5.

[41] E. S. Dana: *System of Mineralogy*, p. 290.

[42] *Annale der Chemie und Pharmacie*, vol. 53, pp. 147 to 156, 1845.

[43] Lleras Codazzi: *op. cit.*, p. 6.

[44] Determined by the method of Wright and Larsen, *American Journal of Science*, vol. 28, p. 423, 1909.

[45] Noted by Scheib oral communication; by Lleras Codazzi, *op. cit.*, p. 5; by W. Reiss and A. Stubel: Geologische Studien in der Republik Colombia, *Reisen in Süd-Amerika*, Berlin, vol. 2, p. 47, 1899; and by Hubert, *op. cit.*, p. 202.

[46] Lleras Codazzi: *loc. cit.*

slender crystals. Presumably a weathering mineral, but Lleras Codazzi[47] noted inclusions of parisite, and Olden[48] mentioned green gypsum as an associate of the emerald.

Albite.—Occurs as a conspicuous component of the albite rock and as small glassy crystals rich in forms[49] in druses in this rock. Found also in one phase of the *Cenicero* and as microscopic crystals in the *Cambiado.*

Barite.—Occurs in conspicuous layers, up to 40 cm. or so in thickness, forming in places the uppermost part of the *Cenicero;* this phase is nodular to massive.[50] Found also as small, glassy, tabular crystals, 2 mm. across, associated with crystallized calcite in some veins in the emerald formation. A small crystal perched on an emerald crystal has been noted by Scheibe.

Anthracite.—Small fragments of impure anthracitic to graphitic carbon are found in joints in the emerald formation.[51]

Marcasite.—Noted by Lleras Codazzi[52] in the form of nodules.

Chalcopyrite.—A few imperfect crystals, stained with a little malachite and azurite, have been found in the workings. Also occurs sparingly in the *Cenicero.*

Native Sulphur.—Found in conspicuous masses in parts of the *Cenicero.*

Pyrophyllite.—Occurs as apple-green folia, occupying small, inconspicuous seams in the emerald formation; found either alone or associated with dolomite, pyrite, and quartz.

Allophanite.—Found only locally developed in clay-like masses forming lenses in the emerald formation. Noted by Lleras Codazzi[53] in blue masses in a vein in the *Cambiado.*

Fuchsite.—Noted by Lleras Codazzi[54] as green laminæ adhering in places to the surface of the shale of the emerald formation.

Age

The ages of the emerald formation and *Cambiado* are fixed as Cretaceous by the fossils, chiefly ammonites, that have been found rather abundantly in them. Miguel Gutiérrez[55] places the *Cambiado* as lower

[47] *Op. cit.*, p. 5.

[48] *Op. cit.*, p. 196.

[49] Six forms were noted by Hubert, *op. cit.*, p. 203.

[50] It is notable that such a conspicuous mineral remained undiscovered until the visit of Scheibe in 1915.

[51] A specimen was given the writer by Dr. Lleras Codazzi.

[52] *Op. cit.*, p. 5.

[53] *Op. cit.*, p. 5.

[54] *Op. cit.*, p. 5.

[55] Miguel Gutiérrez: Estudio Geológico de las Minas de Esmeraldas de Muzo, *Anales de Ingeniería*, pp. 5 to 8, Bogotá, 1913.

Cretaceous[56] and the emerald formation as middle Cretaceous.[57] The present writer presents no fossil evidence but feels that further paleontological study is needed before a correlation closer than "lower Cretaceous" can be accepted for the rocks of the emerald deposits. An ammonite collected by the writer from the stream bed below the workings has been identified by Dr. T. W. Stanton as *Pulchellia zaleatoides*, Karsten, from the upper part of the lower Cretaceous.

Origin

The evidence bearing on the origin of the emerald has been presented in descriptive form. It may be summarized under four heads, as follows:

1. The association of such minerals as emerald, parisite, fluorite, apatite, albite, and barite in a sedimentary formation implies the introduction of material from an external source. This is so obvious from the composition of these minerals and their known occurrence elsewhere as to render further elaboration unnecessary.

2. The presence of pegmatites is significant, because the conditions under which pegmatites form are fairly definitely understood. The mineral content of the pegmatites is thought to correlate their formation with the general period of mineralization.

3. The presence of albite rock (highly albitized limestone) and its spatial relation to a zone occupied by the *Cenicero* and *Cama* indicate the passage of strongly effective mineralizing solutions. The albite rock itself is thought to represent a contact rock, not of the normal type (because of the absence of such characteristic minerals as garnet, epidote, pyroxene, amphibole, etc.) but of the type characterized by V. M. Goldschmidt[58] as that due to "pneumatolitic contact metamorphism," a type that develops later in the cooling of, and more distant from, the parent magma than the normal type.

4. Structural conditions indicate that the emerald formation was overthrust to its present position upon the *Cambiado*, and that this movement was followed by a period of mineralization which attained its most conspicuous results along the fault plane and its economic results above (and not below) that plane. That the emerald veins are the result of the same period of mineralization that produced the *Cenicero*, *Cama*, and albite rock, is thought to be clearly indicated by the mineral content

[56] On the basis of *Crioceras Roemeri*
Ammonites compressissimus, D'Orb.

[57] On the basis of *Hoplites auritus*, Sow.
H. Deshayesi, Leym.
H. tardefurcatus? Leym.
Schloembachia cristata, Defr. sp.
Inoceramus Cuvieri, Sow.

[58] Leith and Mead: *Metamorphic Geology*, pp. 143 to 144, 1915.

and spatial connection that may be traced between the four. The barren calcite veins in the *Cambiado* are probably of the same period of mineralization also; for they are post-faulting (Figs. 8*a* and 8*b*) and in places are connected with the *Cama*.

These considerations together present practically conclusive evidence that the emerald is one effect of a period of mineralization growing out of the intrusion of a body of igneous rock. That exposures of this rock have not been thus far discovered should have little weight as evidence. We may infer further that the emerald was deposited under gas-aqueous (pneumatolitic) conditions, although the general temperature of mineralization throughout was probably below 575°.

Other inferences may be drawn and suggestions made. It is possible that the overthrusting and folding of the emerald formation is due to the crowding effect of the igneous intrusion; this makes an attractive and reasonable hypothesis. Then, the fact that the veins of the emerald formation carry emeralds, while those of the *Cambiado* are barren, or, in short, that the emeralds all lie above the plane of overthrust, although non-economic mineralization proceeded below, suggests that the solutions, entering along the shattered fault plane, effected a separation there, their liquid portion permeating the rocks on either side, their gaseous portions rising and therefore recording passage (by emerald deposition) only in the rocks above. The presence of the two unusual types of deposits, the *Cenicero* and *Cama*, raises a difficult question; but it seems probable that the *Cenicero* was first deposited, following close on to the faulting movement, and then the *Cama* was introduced and, accompanying it (farther out from the fault plane), the calcite veins were developed. Again the carbon content of the emerald formation interposes itself as a common factor, suggesting the possibility that it may have been essential to the formation of the emerald in some way, either by its precipitative action, or by its reducing action on chromium, the coloring agent of emerald. Finally, the question arises as to the source of the calcite so prominent in the seams and the veins throughout, and it appears probable that the calcium carbonate displaced from the *Cambiado* upon its albitization is sufficient to form these bodies, without magmatic contribution of that material.

Mining Methods[59]

The emerald is won exclusively by open-cut mining. The steep slopes of the emerald formation, stripped of their covering of jungle, are worked in great terraced banks (*bancos*), affording benches[60] on which lines of

[59] This subject is treated in some detail by Olden: Emeralds: Their Mode of Occurrence and Methods of Mining and Extraction in Colombia, *Transactions of the Institute of Mining and Metallurgy*, vol. 21, pp. 193 to 209, 1911. The mines were not in operation at the time of the writer's visit.

[60] The benches are about 76 cm. high and 76 cm. broad.

peons stand and attack the bench below with long iron crowbars (Figs. 11 and 12). The comparatively soft limestone and shale are easily broken away in this way without recourse to blasting (which would

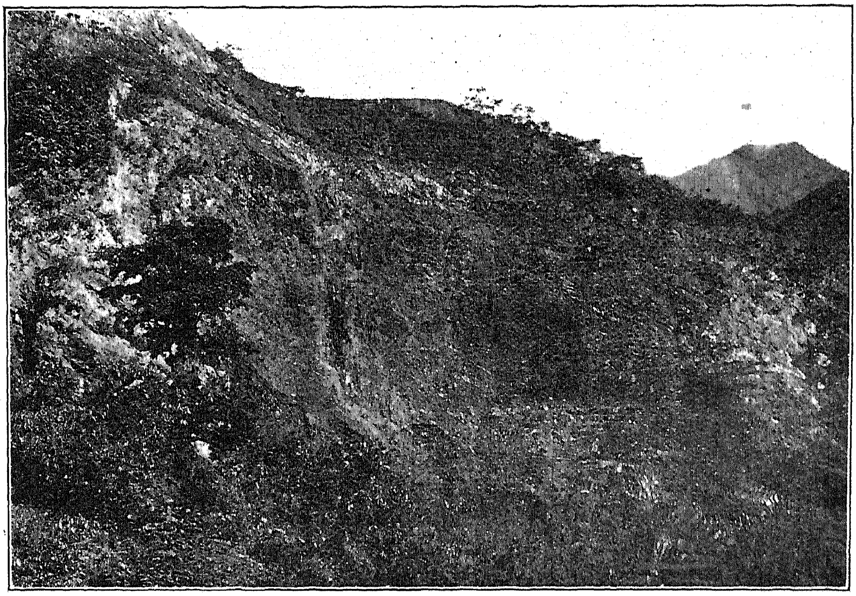

FIG. 11.—PHOTOGRAPH OF BANCO CENTRAL, ONE OF THE PRINCIPAL OPEN CUTS, SHOWING THE TERRACES BY MEANS OF WHICH THE EMERALD FORMATION IS WORKED.

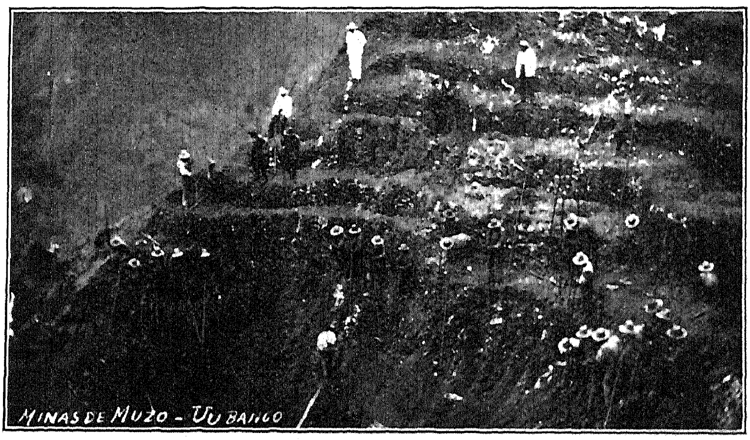

FIG. 12.—PHOTOGRAPH SHOWING "PEONS" WORKING ON A BANK, BREAKING DOWN THE EMERALD FORMATION WITH LONG, IRON CROWBARS.

shatter the fragile emerald crystals) and the emerald-bearing calcite veins are carefully removed by hand and taken to a sorting shed above. The débris falls down the step-like slope and the accumulation at intervals

is swept down to the creek below by water led from reservoirs in the mountains above the workings (Fig. 13).

In the sorting shed, the calcite veins are carefully broken by hand and the emerald crystals picked out. The finer material, together with gem-bearing débris gathered from "bed-rock" and from the water channels below the banks, is washed on sloping tables and the emerald fragments withdrawn by boys (Fig. 14). The stones are separated into a number of grades according to color, size, transparency, and freedom from flaws.

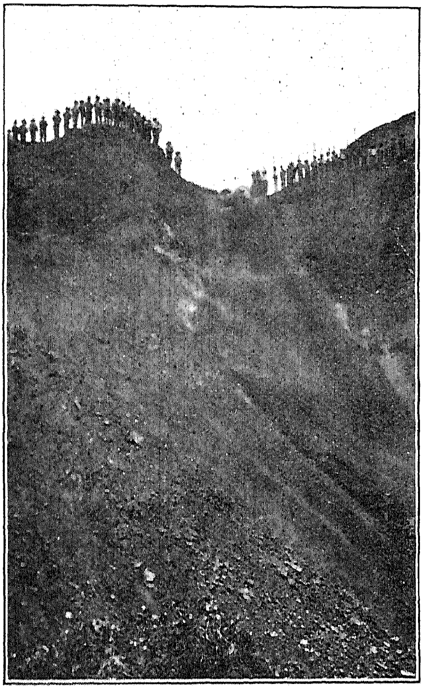

Fig. 13.—Photograph Showing How Accumulated Débris is Cleared from Working Face by Water Let in from Above.

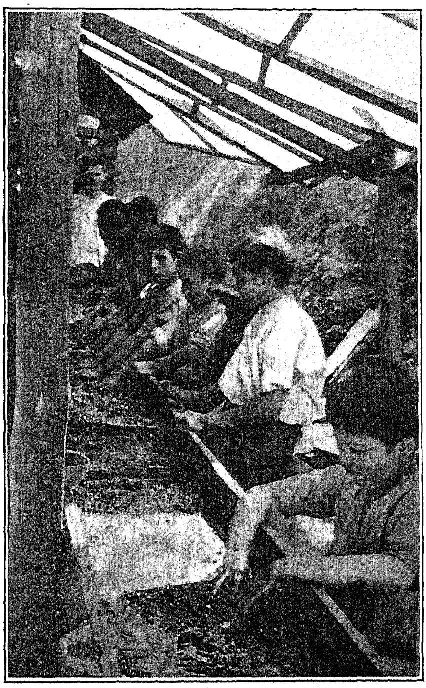

Fig. 14.—Photograph Showing Sorting Shed and Manner in Which Emeralds are Separated from the Crushed Matrix.

The product goes by mule to Bogotá from time to time, and there awaits transportation to London in larger consignments.

The labor is done by Indian *peons* drawn from the neighborhood. Mining officials and police are supplied from Bogotá or other towns. Great vigilance is exercised, when the mines are in operation, to reduce loss by theft. A body of military police is assigned to the mines; the exits are carefully guarded; watchmen are always on duty in small guard-houses on prominent points above the workings; overseers are in constant attendance during hours of work; and the workmen are impounded

and not allowed to leave the mines until the culmination of a suitable period of search.

The mine buildings are commodious and comfortable, maintained in good condition (Fig. 3). The mining equipment is simple, but the fragility of the emerald precludes the use of most types of equipment that would increase the quantity of ground handled.

Production

It is impossible to present an approximation of the total production of the Muzo mines. The pre-Spanish output, undoubtedly significant, is of course not open to any measure. In historic times, the exploitation was so irregular, and the records so incomplete, that a fair basis for judgment is entirely lacking. Nevertheless, it is certain that the total output may be estimated in terms of tens of millions of dollars, and that in many single years the production has run in value from $1,000,000 to perhaps $2,000,000, or more.

Other Deposits

The Coscuez and Somondoco emerald deposits have already been mentioned as the only other important known occurrences of this mineral in South America.

Coscuez Deposits.—These lie about 12 km. in direct line north-northwest of the Muzo mines (Fig. 2) but are exceedingly difficult of access. They were known before the Conquest and won a reputation for richness (see p. 914). No information is yet available concerning their geology, but the writer has been informed that a geological study of them was made late in 1915 by Robert Scheibe.[61]

Somondoco Deposits.—These lie about 130 km. in direct line southwest of the Muzo mines on the Orinoco watershed (Fig. 1), and a careful survey of available information suggests that possibly they are richer than the Muzo deposits. They have been visited and described by W. Lidstone[62] and by E. B. Latham;[63] and in 1915 a detailed geological survey of them was made by Robert Scheibe, but the results are not yet published.

These deposits have a romantic history. They were richly productive before the Spanish Conquest, were seized and worked a while by the Spaniards, were subsequently abandoned and lost, and only rediscovered in 1896.[64] They have not been productive in recent times, though foreign capital has interested itself in their exploitation.

[61] Personal communication from Dr. Juan de Dios Vasquez, Director of the Muzo mines.

[62] Emerald Mines of Chivor, Colombia, manuscript, Bogotá, 1906, copy on file, U. S. Geological Survey, Washington.

[63] The Newly Discovered Emerald Mines of "Somondoco," *School of Mines Quarterly*, Columbia Univ., vol. 32, pp. 210 to 214, 1910–11.

[64] Latham: *op. cit.*, p. 211.

According to Latham[65] the emerald here occurs in veins of "semi-decomposed quartz" traversing folded beds of dark gray to black "clay-slate" and limestone, and is found either directly embedded, or in pockets, in these veins, very rarely in the rock itself. Lidstone[66] describes the occurrence here in a similar manner, without, however, specifying the vein matter to be quartz.

Acknowledgments

For courtesies and valuable help, both during the writer's visit to the Muzo mines, and later during the preparation of this paper, the writer extends his appreciative acknowledgment to the following: Hon. Marco Fidel Suárez, Minister of Foreign Affairs, Bogotá; Hon. Daniel J. Reyes, Minister of *Hacienda*, Bogotá; Hon. Thaddeus A. Thompson, American Minister, Bogotá; Dr. Juan de Dios Vasquez, Director of the Muzo mines; Dr. Alfredo Angueyra, Acting-Director of the Muzo mines at the time of the writer's visit; Prof. Dr. Robert Scheibe, Professor of Geology, Royal Mining Academy, Berlin; Dr. Ricardo Lleras Codazzi, Professor of Mineralogy and Geology, National University, Bogotá; Dr. Lucas Caballero, Bogotá; Dr. Hermano Apolinar María, Bogota; Phanor James Eder, New York; Douglas B. Sterrett, U. S. Geological Survey, Washington; Dr. George P. Merrill, and Dr. Edgar T. Wherry, U. S. National Museum, Washington; Dr. Ronald S. Crane, Evanston, and Mrs. Leonard G. Shepard, Evanston.

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Ricardo Lleras Codazzi: Los minerales de Muzo, *Contribución al estudio de los minerales de Colombia*, Imprenta de la Cruzada, pp. 3 to 7, Bogotá (1915). Good, brief description of the Muzo minerals.

Charles Olden: Emeralds: Their Mode of Occurrence and Methods of Mining and Extraction in Colombia, *Transactions of the Institute of Mining and Metallurgy*, vol. 21, pp. 193 to 209 (1911). Good account of mining methods at Muzo. Includes geological description and historical details, the latter in part unreliable.

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[65] *Op. cit.*, p. 212.

[66] *Op. cit.*

Discussion

Edgar T. Wherry,* Washington, D. C. (communication to the Secretary†).—Dr. Pogue's presentation of the facts concerning the emerald deposits is very clear and convincing, and the only addition that I can suggest is a summary of previous theories of origin. He makes it evident that the pegmatite theory is the only one capable of explaining the existing relations, but upon certain details there may be some difference of opinion. If I understand the term pneumatolytic, it does not imply that all the elements concerned in a given deposit were transported as gases, but rather that the crystallization of these elements into the various minerals was favored by the presence of certain gaseous substances, notably H_2O, CO_2 and HF. It is highly improbable that the oxides of glucinum, aluminum, chromium, and silicon, which enter into the composition of the mineral emerald could have been transported in the gaseous form. The same is true of the metallic constituents of the parisite and other associated minerals. The explanation suggested, that solutions separated into liquid and gaseous portions, the latter ascending and forming the emerald in the upper portions of the rock only, therefore, seems to me untenable.

When two formations exist side by side and one, *A*, is mineralized while the other, *B*, is barren, the possible explanations may be classed as (1) chemical, and (2) physical.

1. Some chemical feature of *A* not found in *B* might have caused crystallization of certain minerals in the former, which did not appear in the latter. In the present instance both rocks appear to be so similar chemically that no such effect can be looked for. Pogue mentions carbon as a possible precipitating agent, but describes both formations as carbonaceous, so that the difference in the minerals of the two can not be thus explained.

2. The physical condition of *A* might have permitted or encouraged the passage of the solutions, while that of *B* retarded or prevented it. In the present deposit some mineralization occurs in both formations, calcite veins, albite, and pyrite being found in both, whereas emerald, parisite, and a number of minerals of minor importance occur only in the upper, *A*. It seems to me that this difference may have been produced by a change in the composition of the solutions during the progress of mineralization; at first these brought in only the constituents of albite and pyrite and deposited them with calcite dissolved from the wall rock, in both formations; the openings in *B* became completely filled, while the more numerous or larger ones which would naturally have developed in *A*, since it was the uppermost formation, remained partially open. Then, when during later phases of the mineralizing activity the constituents of emerald and parisite appeared, they were deposited only in *A* because *B* had become impermeable.

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The Radio-Activity of Allanite

BY L. S. PRATT,* CHARLOTTESVILLE, VA.

(Arizona Meeting, September, 1916)

In 1910 the author was engaged in a qualitative study of the radio-activity of several chemical substances and a few minerals. In the course of the work he studied the mineral allanite (obtained from granite ledges at Topsham, Maine), and found it to be appreciably radio-active.

Recently the fact was brought to his attention that the radio-activity of allanite is not well known, although Hon. R. J. Strutt in 1905 published a list of radio-active minerals with their analyses, including allanite from Amherst County, Virginia.[1]

In the work done, a standard type of electroscope was used, and the readings were made with a telescope, fitted with a scale, by means of which the rate of discharge could be readily determined.

The natural "leak" of the instrument was carefully determined, and all results corrected accordingly.

The method of study was as follows: The material to be tested was finely powdered, and spread over the bottom of a small, inactive pan. The electroscope was then charged, and the pan placed in position. The door of the instrument was closed, and the rate of discharge carefully determined.

All results were reduced to common form to permit of comparison.

It is to be noted that the materials tested were not used in strictly comparable quantities, such as if 1 g. of each had been used, but about the same quantity of powder was used (in the cases of the solid substances), so that the results are qualitatively accurate.

The results obtained were the following, reduced to common form and corrected for the natural "leak" of the instrument used:

	Mm. in 6 Min. (on Scale)
"Leak"	1
Radium residue (in pan)	720
Thorianite	62
Nernst mantle	17
Uranium nitrate	11
Allanite	3

* University of Virginia.

[1] *Proceedings of the Royal Society of London*, Ser. A, vol. 76, pp. 88 to 102, 312 (1905). Abstract in *Zeitschrift für Krystallographie*, vol. 43, p. 610 (1905), and *Chemical News*, vol. 91, p. 299 (1905).

Zircon-Bearing Pegmatites in Virginia

BY THOMAS L. WATSON,* PH. D., CHARLOTTESVILLE, VA.

(Arizona Meeting, September, 1916)

Introduction

THE occurrence of zircon in pegmatites of acidic composition is recorded by many observers both in this country and abroad, and they form one of the most important geologic modes of occurrence of the mineral.[1] As is well known, commercially the most important American locality for zircon is near Zirconia in Henderson County, North Carolina, where many tons of the mineral have been obtained from a kaolinized pegmatite dike 100 ft. wide and traced for 1½ miles along the direction of strike N. 50° E.

Zircon may be developed in pegmatites as inclusions in the principal rock-forming minerals, chiefly quartz and feldspar, and as a separate megascopic constituent in the form of grains and crystals not exceeding, as a rule, 2 in. in size, and usually smaller. In the Henderson County, North Carolina, pegmatite, zircon occurs in prismatic crystals with pyramidal terminations measuring up to 30 mm. in diameter, associated chiefly with the feldspar. In some of the apatite veins of Canada, which are closely allied to pegmatite dikes, zircon is reported in crystals upward of 6 in. in length and 2 in. and more in thickness.

Zircon in large masses occurs in Virginia in the well-known pegmatites near Amelia in Amelia County, and near Gouldin in Hanover County. Both localities are in the middle eastern portion of the Piedmont Plateau province, and are separated by a distance of about 40 miles in a northeast-southwest direction. The rocks of both areas, including the pegmatite bodies, are in an advanced stage of decay, and the hard and moderately fresh rocks are concealed beneath a cover of variable thickness of rock decay derived by the normal processes of weathering. Exposures, there-

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[1] For a summary of the properties, occurrence, and uses of zircon, see paper by T. L. Watson and F. L. Hess on Zirconiferous Sandstone near Ashland, Virginia, in *Bulletin of Philosophical Society of the University of Virginia*, Scientific Section, vol. 1, No. 11, pp. 267–292. (1912.)

fore, of even moderately fresh rock in the two areas are rare except along stream courses.

Pegmatite dikes occur in most of the Virginia Piedmont counties, and the detailed study of them now in progress by the State Geological Survey will doubtless add to those of Amelia and Hanover Counties that are zircon-bearing. The particularly interesting features of the zircon noted in the pegmatites of the two Virginia localities are (1) its occurrence in massive forms of unusual size, and (2) its association in the two localities with an entirely different group of rarer minerals, although the pegmatites of each area are of granitic composition. Zircon and apatite are the only rare minerals that have been found alike in the two areas, and, so far as the pegmatite bodies have been worked, zircon is more abundant in the pegmatites of the Hanover County area than in those of Amelia County.

Amelia County Area

The pegmatite bodies occurring near Amelia Courthouse in Amelia County have long been known for the variety of rare minerals found in them, many of which were of unusual size. The dikes have been worked from time to time for a long period of years as a source of commercial mica and feldspar, and to a less extent of minerals for the gem trade.

The country rock is a thinly foliated, moderately dark-colored, fine-grained, biotite gneiss or schist, containing more or less muscovite. Where measured, the foliation strikes N. 25° to 30° E. and dips 40° to 50° N. W. Diabase dikes of Mesozoic age intrude the rocks in places. The pegmatite bodies are dike-like in form and nearly vertical, with the direction of trend doubtful. They cut across the foliation of the schists and the large ones will measure more than 50 ft. across. They are cut by joints, but there is no evidence of schistose structure developed from metamorphism.

The pegmatites are of granitic (acidic) composition, containing feldspar, including the potash varieties, orthoclase and green microcline, and the soda variety, albite, with quartz and muscovite, and a large number of rarer minerals. The principal rock-forming minerals are not uniformly distributed through all parts of the pegmatites, but their distribution is very irregular, first one and then another of these minerals predominating in different parts. The albite, occurring in splendid crystallizations as reticulated platy forms of bluish white to white color and frequently transparent, is of a high degree of purity as indicated in the two analyses on the following page.

The texture of the pegmatites is granular consertal rather than graphic. Miarolitic cavities have been observed in some of the openings made in the pegmatites. One of these was of large size, the walls of

which were lined with crystals of smoky quartz and pure white crystals of albite, some as transparent as glass.[2]

Analyses of Albite from Amelia County, Virginia

	I	II
SiO_2	68 44	68.22
Al_2O_3	19 35	19.06
Fe_2O_3	..	0.15
CaO	. .	0.40
Na_2O	11 67	11.47
K_2O	0.43	0.20
H_2O	.....	0.69
	99.89	100.19
Specific gravity	2.605	

I. R. N. Musgrave, *Chemical News*, vol. 46, p. 204 (Nov. 3, 1882).
II. E. T. Allen, *Bulletin No.* 591 *of U. S. Geological Survey*, p. 300 (1915).

The rarer minerals include representatives of five distinct chemical groups: (1) *Haloids*, including fluorite; (2) *silicates*, including garnet (spessartite), black tourmaline, beryl, helvite, allanite, and zircon; (3) *niobates*, including columbite; (4) *tantalates*, including microlite; and (5) *phosphates*, including apatite and monazite. With the exception of fluorite, tourmaline, and zircon, each of the minerals has been analyzed with the results shown below. Some of these minerals have been found only occasionally in the Amelia pegmatites and are very rare. Many of them attained unusual size, such as crystals of beryl 3 to 4 ft. long and 18 in. thick, columbite in crystalline masses weighing 6 to 8 lb., allanite crystals more than 15 in. long, microlite in masses up to 8 lb. in weight, and monazite in masses larger than those of microlite.[3] Zircon has been noted in small crystals and in masses weighing several pounds. Stibnite and galena have been reported, but they are extremely rare and have not been seen by the writer.

Among the rarer minerals found in the Amelia County pegmatites, analyses of the garnet, beryl, helvite, allanite, columbite, microlite, apatite, and monazite have been made and are given below in the order named. The general character and mode of occurrence of these minerals in the Amelia County pegmatites have been fully described by Professor Fontaine.[4] No analysis has been made of the zircon. It is much less abundant than in the pegmatites of the Hanover County area and properly belongs to the more sparingly occurring rare minerals in the Amelia County area.

[2] W. F. Fontaine: Notes on the Occurrence of Certain Minerals in Amelia County, Virginia, *American Journal of Science*, Series 3, vol. 25, p. 332 (1883).
[3] W. F. Fontaine, *ibid.*, pp. 330–339 (1883).
[4] *Ibid.*, pp. 330–339 (1883).

	Spessartite			Beryl
	I	II		III
SiO_2	36.34	35 35	SiO_2	65 24
Al_2O_3	12 63	20 41	Al_2O_3	17.05
Fe_2O_3	.....	2.75	Fe_2O_3	2.20
FeO	4.57	1 75	BeO	12.64
MnO	44.20	38 70	CaO	0.57
MgO	0.47	None	Na_2O	0.68
CaO	1.49	0 94	H_2O	2.70
Ign.	Trace	0 27		
				101.08
	99 70	100 17		
Specific gravity	4.20	.....	Specific gravity	2.702

I. C. M. Bradbury: *Chemical News*, vol. 50, p. 220 (Nov. 7, 1884). See also W. H. Seamon, *Chemical News*, vol. 46, p. 195 (Oct. 27, 1882).

II. F. W. Clarke: *Bulletin No. 60 of U. S. Geological Survey*, p. 129 (1890).

III. R. W. Barker: Analysis of a Beryl from Amelia C. H., Amelia Co., Virginia, *American Chemical Journal*, vol. 7, No. 3, pp. 175–176 (Oct., 1885).

	Helvite			Allanite
	IV	V		VI
SiO_2	25.48	31.42	SiO_2	32.35
BeO	12.63	10.97	Al_2O_3	16.42
MnO	39.07	40.56	Fe_2O_3	4.49
FeO	2.26	2.99	Ce_2O_3	11.14
Mn	8.66	8.59	Di_2O_3	6.91
S	4.96	4.90	La_2O_3	3.47
Al_2O_3	2 95	0.36	FeO	10.48
CaO	0.71	.....	MnO	1.12
Na_2O	1.01	.....	CaO	11.47
K_2O	0.43	.....	$\left.\begin{array}{l}Na_2O\\K_2O\end{array}\right\}$	0.46
	98.16	99.79	H_2O	2 31
				100.62
Specific gravity	....	3.25	Specific gravity	3.323

IV. R. Haines: *Proceedings of Academy of Natural Sciences of Philadelphia*, 1882 p. 101.

V. B. E. Sloan: *Chemical News*, vol. 46, p. 195 (Oct. 27, 1882).

VI. F. P. Dunnington: *Proceedings of Academy of Natural Sciences of Philadelphia*, 1882, p. 103.

	Columbite VII		Microlite VIII
Nb_2O_5	31.40	Ta_2O_5	68.43
Ta_2O_5	53.41	Nb_2O_5	7.74
SnO_2	Trace	WO_3	0.30
FeO	5.07	SnO_2	1.05
MnO	8.05	CaO	11.80
CaO	1.27	MgO	1.01
MgO	0.20	BeO	0.34
Y_2O_3(?)	0.82	UO_3	1.59
		Y_2O_3	0.23
	100.22	$(Ce, Di)_2O_3$	0.17
Specific gravity	6.48	Fe_2O_3	0.29
		Al_2O_3	0.13
		Na_2O	2.86
		K_2O	0.29
		F	2.85
		H_2O	1.17
			100.25
		Specific gravity	5.656

VII. F. P. Dunnington: *American Chemical Journal*, vol. 4, No. 2, p. 138 (May, 1882).
VIII. F. P. Dunnington: *American Chemical Journal*, vol. 3, No. 2, pp. 130–133 (May, 1881).

	Apatite IX		Monazite X	Monazite XI
P_2O_5	41.06	P_2O_5	24.04	26.12
CaO	53.94	Ce_2O_3	16.30	29.89
F	3.30	La_2O_3	10.30	26.66
Cl	Trace	Di_2O_3	24.40	
Al_2O_3	0.19	$(Y, Er)_2O_3$	1.10	.....
Fe_2O_3	0.81	SiO_2	2.70	2.85
Insol.	0.63	ThO_2	18.60	14.23
Ign.	0.81	Fe_2O_3	0.90	.....
		Al_2O_3	0.04	.....
	100.74	Ign.	.....	0.67
Specific gravity	3.161			
			98.38	100.42
		Specific gravity	.....	5.30

IX. G. H. Rowan: *Chemical News*, vol. 50, p. 208 (Oct. 31, 1884).
X. F. P. Dunnington: *American Chemical Journal*, vol. 4, No. 2, p. 140 (May, 1882).
XI. S. L. Penfield: *American Journal Science*, vol. 24, pp. 250–254 (1882).

Hanover County Area

The zircon-bearing pegmatites of Hanover County form a part of the recently discovered but fairly well-known rutile area of Goochland and Hanover Counties,[5] which lies about 25 miles northwest of Richmond (see map, Fig. 1). The principal rock of the region is a gneiss of variable composition, chiefly micaceous (biotite and muscovite) and at times

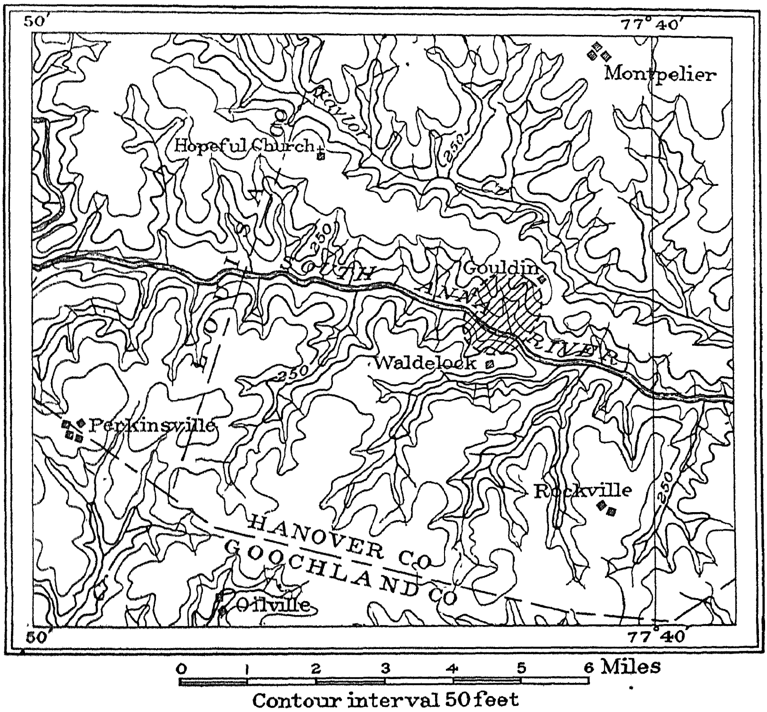

FIG. 1.—TOPOGRAPHIC MAP OF A PART OF HANOVER COUNTY, VIRGINIA, SHOWING LOCATION OF GOULDIN ZIRCON-BEARING PEGMATITE AREA. (BASED ON GOOCHLAND SHEET OF U. S. GEOLOGICAL SURVEY.)

hornblendic, cut by numerous pegmatites, some of which are rutile-bearing, and a variety of basic igneous rocks. Microscopic study of thin sections of the gneiss shows it to conform in composition to an original acidic igneous rock of the granite type. The banded structure is secondary, developed by regional metamorphism.

The pegmatite bodies range up to 4 ft. and more in width, are of light color and granitic composition, and lie mostly in the foliation planes of the gneiss. Like the inclosing gneiss, they have been mashed and

[5] T. L. Watson and S. Taber: Geology of the Titanium and Apatite Deposits of Virginia, *Virginia Geological Survey Bulletin*, III A, pp. 248–261 (1913).

squeezed from metamorphism and are schistose in structure. Much of the feldspar has been granulated and the quartz mashed into lens-like masses. The pegmatites are dominantly feldspathic, containing both potash and soda-lime varieties, with some quartz, and less muscovite and biotite. Neither amphiboles nor pyroxenes occur in the pegmatites of the Hanover or Amelia County areas. Rutile in small grains and masses up to 15 and 20 lb. and more in weight[6] is an important mineral in some of the Hanover County pegmatites, and is found over parts of the area in such quantity as to be of commercial value.

The rutile is an original constituent of the pegmatites and much of it shows granulation and mashing like the fieldspar and quartz. Besides the usual rock-forming minerals of the pegmatites, the rarer minerals are exceptionally few in number. Rutile, the most abundant and important one, is associated with ilmenite which may occur as separate grains and masses or as an intimate mixture or intergrowth with rutile. Occasional apatite has been found, and in places indications of the former presence of pyrite.

In addition to these, zircon has recently been found associated with rutile in the pegmatites near Gouldin in the Hanover portion of the rutile area. (See map, Fig. 1.) The mineral has been found in irregular fragments and masses up to about 2 lb. in weight. One of the larger masses examined by the writer appears to have been broken from a large crystal of the mineral. Like the other constituents of the pegmatites, every specimen of the zircon studied shows mashing and squeezing from metamorphism. The color is irregular even in the same mass, ranging from reddish-brown through grayish to colorless. Although a chemical analysis of the zircon has not been made, laboratory tests carried out on a number of pieces of the mineral show it to be quite pure. The many pieces of the mineral found on the surface, due to the extensive weathering of the pegmatite bodies, encourage the belief that the mineral is by no means a rare constituent of the dikes in this area, and may be found in quantity to be of commercial value.

[6] F. L. Hess mentions one mass of rutile that was reported to have weighed 200 to 300 lb. *Mining World*, vol. 33, pp. 305–307 (Aug. 20, 1910).

Iron Pyrites Deposits in Southeastern Ontario, Canada

BY P. E. HOPKINS,* B. A., B. S., TORONTO, ONT.

(Arizona Meeting, September, 1916)

Introduction and History

IN speaking of the economic geology of southeastern Ontario, W. G. Miller and C. W. Knight[1] say that "there occurs in southeastern Ontario a variety of minerals and rocks of economic value, probably as great as in any district of like size on the North American continent. Some of these deposits, including marble and trap, are inexhaustible. Others, including talc and iron pyrites, have proved to be of considerable economic importance. From time to time, during the last 50 years, the following minerals and rocks have been mined or quarried with varying success: gold, iron pyrites, zinc blende, copper pyrites, galena, mispickel, magnetite, hematite, talc, actinolite, mica, marble, ophicalcite, feldspar, fluorite, apatite, corundum, graphite and sodalite. All of the economic materials, with the exception of fluorite, appear to be of pre-Cambrian age. The fluorite veins penetrate the Ordovician, Black River, limestone."

Accompanying that report was an article by the writer on the Queensboro Pyrite Area which includes one of the two working pyrite properties in southeastern Ontario.

In the present paper will be given a brief description[2] of all the known pyrite deposits in the area which may at some time possess an economic value, with fuller descriptions of the two working mines—The Canadian Sulphur Ore Co.'s mine near Queensboro, and the Nichols Chemical Co.'s property at Sulphide.

The earliest mining of iron pyrites in Ontario was done in 1868 on the Billings property near Brockville. The mines were closed down in 1879 under the assumption that they were exhausted. Many other

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[1] The Pre-Cambrian Geology of Southeastern Ontario, *Annual Report, Ontario Bureau of Mines*, vol. 22, part 2 (1913).

[2] The information regarding the various pyrite prospects in southeastern Ontario is summarized from E. L. Fraleck's comprehensive report on Iron Pyrites in Ontario, *Annual Report, Ontario Bureau of Mines*, vol. 16, part I, pp. 149–201 (1907).

pyrite deposits have been worked for gold, iron or copper at some time. The steady pyrite industry of the Province began in 1900 when ore from the Bannockburn mine was produced. Mines in Hastings County have been steady producers since that time. An acid-making plant has been in operation at Sulphide since 1907 by the Nichols Chemical Co. for the

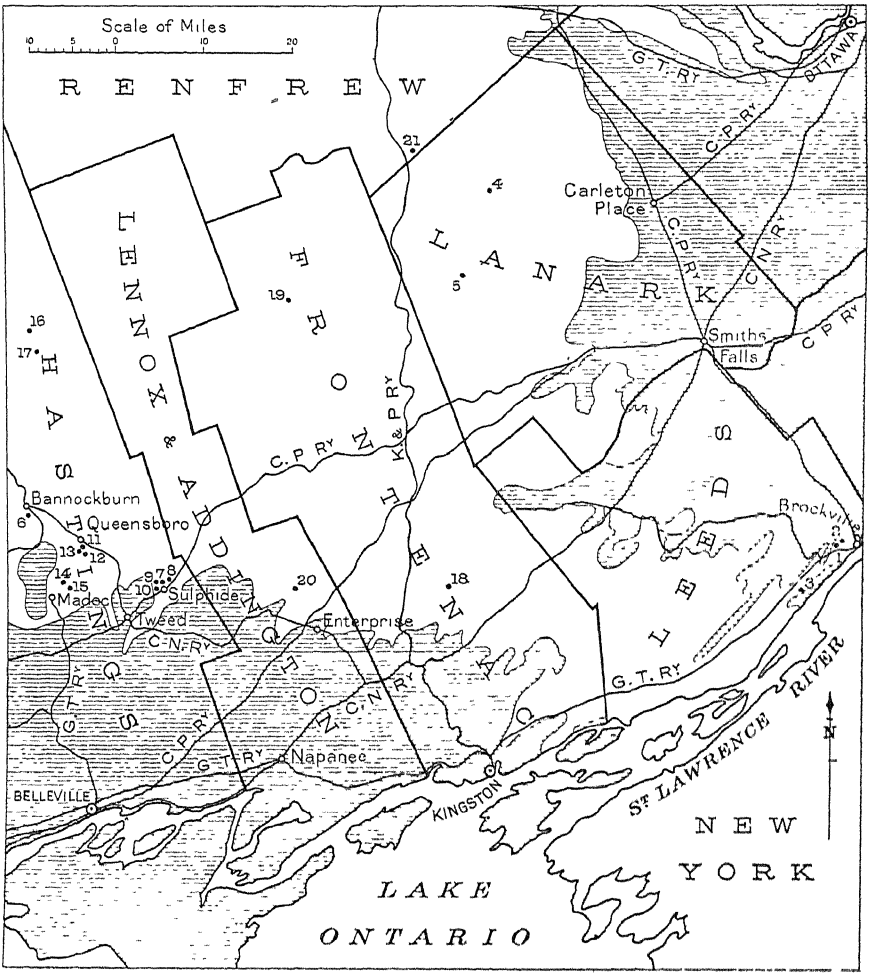

Fig. 1.—Sketch Map Showing Pre-Cambrian and Paleozoic (Hatched) Areas in Southeastern Ontario. The Numbers Indicate the Locations of Properties Described.

treatment of its ore at Sulphide. The company also buys the ore mined from other properties in the neighborhood. Another plant for treating custom ore is operated by the Grasseli Chemical Co. at Hamilton. These two plants treat the bulk of the eastern Ontario production, the remainder being shipped to the United States.

Recently a large percentage of the production has been coming from the Vermilion Lake deposits[3] in northwestern Ontario, the ore being shipped to United States ports on the great lakes. Another property, the Goudreau Lake deposits,[4] has been recently developed and expects to

TABLE 1.—*List Showing the Locations of Pyrite Deposits in Southeastern Ontario*[5]

Brockville Section.
- † 1. Brockville Chemical Co. (Billings property); lot 19, con. 2, Elizabethtown township.
- † 2. Sloan prospect; lot 18, con. 2, Elizabethtown township.
- 3. Shipman prospect; about 6 miles west of the Billings (No. 1).

Lanark County.
- † 4. McIlwraith mine; lot 5, con. 4, Darling township.
- 5. Ladore prospect; lot 19; con. 7, Dalhousie township.

Hastings County.
- ‡† 6. Bannockburn (or Jarman) mine; lot 25, con. 6, Madoc township.
- * 7. Hungerford mine (Nichols Chemical Co.); lot 23, con. 12, Hungerford township.
- † 8. Canada mine (formerly Oliver Prospect); lot 26, con. 12, Hungerford township.
- 9. Hungerford Western Extension; parts of lots 21 and 22, con. 12, Hungerford township.
- † 10. Ontario Sulphur Mines, Ltd.; northwest quarter of east half of lot 21, con. 12, Hungerford township.
- † 11. Queensboro mine; lot 11, con. 11, Madoc township.
- * 12. Canadian Sulphur Ore Co. (formerly Wellington prospect) N. ½ lot 9, con. 10, Madoc township.
- † 13. Davis or Palmer prospect; lot 10, con. 10, Madoc township.
- 14. Farrell prospect; 2 miles northeast of Madoc village.
- ‡ 15. McKenty prospect; 2 miles east of Madoc village.
- 16. Little Salmon Lake deposit; lot 23, con. 7, Cashel township.
- 17. Gunter property; lot 23, con. 4, Cashel township.

Other Eastern Ontario Prospects.
- 18. Snooks prospect; lot 7, con. 14, Loughborough township, Frontenac Co
- 19. Stalker prospect; lot 42, con. 6, Clarendon township, Frontenac Co.
- 20. Foley prospect; 5½ miles north of Enterprise Sta , Lennox, Addington Co.
- 21. Caldwell prospect; lot 1, con. 1, Blithfield township, Renfrew Co.

* Mines now working (April, 1916) and shipping pyrites.
† Properties which have shipped pyrites.
‡ Properties which have shipped hematite or limonite.

[3] E. S. Moore: Vermilion Lake Pyrite Deposits, *Annual Report, Ontario Bureau of Mines*, vol. 20, part I, pp. 199–209 (1911).

T. F. Sutherland: Northern Pyrites Company, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, pp. 94–95 (1915).

[4] A. L. Parsons: Goudreau Pyrite Claims, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, p. 211 (1915).

T. F. Sutherland: Madoc Mining Company, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, p. 107 (1915).

[5] The number of each property refers to the corresponding number showing its position on the accompanying map.

commence at once supplying large tonnages. The Helen mine,[6] operated by the Algoma Steel Corporation, produces some pyrite which is treated in its plant at Sault Ste. Marie.

The iron pyrites resources of Ontario are of considerable extent and value, in the last 15 years 538,755 tons, worth $1,438,122, having been produced, the greater part coming from southeastern Ontario. During the coming years there will undoubtedly be a steady increase in production. The war has had a stimulating effect on the demand of the United States for pyrite from Ontario.

Brockville Section

The Brockville Chemical Co., No. 1,[7] began mining for pyrite on the Billings property in 1868. The ore occurred in a series of lenses conformable to the lamination of a highly foliated pink granite gneiss. The lenses, which consist of pyrite and calcite in parallel lines, strike northeast and dip to the southeast. The richer shoots of ore were gouged out and no timbering was done. The main pit was sunk 250 ft. The ore was used for making acids in Brockville, the sulphuric and mixed acids being used at the fertilizer and dynamite works in and near Brockville. Operations of all kinds ceased in 1880. The evidence of the men who worked in the old pits is to the effect that they were never completely exhausted.

Sloan Prospect, No. 2.—A band of gossan strikes in a northeast direction across the property and dips to the southeast. The 20-ft. inclined shaft passes through 6 or 8 ft. of gossan. There is a width of 3 ft. of solid pyrites on the foot wall, the remainder of the shaft being in alternating bands of pyrite and crystallized calcite in equal amounts. Eighty tons of ore, running 40 per cent. sulphur, were shipped to Buffalo and Capelton.

The Buffalo-Brockville Mining Co. shipped a small tonnage from this lot during 1911 and 1912.

Shipman Prospect, No. 3.—The pyrite, which is much intermixed with pyrrhotite and country rock (gneiss), has been mined from an irregular pit 40 ft. long and 30 ft. wide.

Lanark County

McIlwraith Mine, No. 4.—The deposit, which is covered by 14 ft. of gossan, strikes north of east along a contact between diorite on the

[6] A. L. Parsons: Helen Mine, *Annual Report, Ontario Bureau of Mines*, vol. 24, part I, pp. 202–205 (1915).

[7] The numbers following mention of the pyrite properties refer to corresponding numbers showing their positions on the accompanying map, and in Table 1.

south and crystalline limestone on the north, and dips 60° to the south. It was first opened for gold. In 1899 and 1900 the shaft was deepened to 75 ft. and a 150-ft. tunnel run along the strike of the deposit, disclosing a length of over 90 ft. of clean high-grade pyrite inclosing lenses of quartz. A 12-ft. crosscut to the south did not pierce the width of the deposit. Three carloads of ore were shipped. Samples from the dump and tunnel, by E. L. Fraleck, gave 38.86 and 42.60 per cent. of sulphur respectively.

Ladore Prospect, No. 5.—A heavy fahlband strikes north of east along the contact of a coarse amphibolite and a fine-grained gray granite. The trenches and shallow pits expose a gossan in the form of bog iron ore, but pyrite in quantity was not located. The fahlband continues into the adjoining lot to the east along a contact of crystalline limestone and granite.

Bannockburn Mine, No. 6.—In 1898, the property was opened as an iron mine, 11 car loads of limonite, running about 38 per cent. in iron and low in sulphur, having been shipped. This ore was merely a gossan 8 to 15 ft. deep which capped iron pyrites deposits. The pyrites occurred as two lenses at right angles to each other, but conforming in strike and dip with the inclosing rock, a chloritic schist. Limestone covers the apex of the fold of the lenses. The south lens, which is 160 ft. long and 8 to 15 ft. wide, was mined to a depth of 275 ft. During the 6 years of operation about 580 tons of pyrite per month were shipped, all of which went to the General Chemical Co. at Buffalo. The ore did not fall off either in grade or quantity with depth, but, owing to the hazard of open-pit mining, operations were abandoned in August, 1906.

Hungerford Mine, No. 7.—This property was opened 40 years ago as a gold property, and a smelter was erected to extract gold from the barren pyrite. The Nichols Chemical Co.[8] re-opened the mine in June, 1903. Owing to some difficulty about the title, the mine was closed down in August, 1904, but operations were resumed in August, 1905, and have since been continuous. Since 1907, acid works have been in operation for the treatment of company ores, and other ores in the vicinity.

Passing through this property, and extending beyond, is a large fahlband striking 25° north of east and traceable for 2 miles. Level farm land to the south is underlain by garnetiferous crystalline schist cut by massive diorite, into which, 500 yd. north of the deposits, has been intruded a pink hornblende granite that rises above the country in a series of rugged hills, locally called the Bald Mountains. The granite has protected the deposits from denudation. The deposits are strung along the contact of the diorite and the schist, the strike of lenses, contact, fahlband, and schist being identical.

The pyrite occurs in three parallel deposits striking with the schist

[8] W. H. Nichols, President, 25 Broad St., New York.

and dipping 60° to the south. The middle one, which does not outcrop on the surface, lies 85 ft. from the south vein and 45 ft. from the north deposit. The north deposit, upon which most of the work has been done, varies in width from 6 to 22 ft. It has been exploited to a length of 620 ft. and to a depth of 575 ft., and the ore still continues. The length as indicated on the surface is about 500 ft. There are now two shafts on the property and about 3,500 ft. of drifting has been done on the orebodies on the six levels.

The ore is coarsely granular and makes a large percentage of fines. The main impurity is calcite, although there is also some quartz present. A small quantity of pyrrhotite occasionally occurs, mainly in the north lode next the foot wall. The average percentage of run of mine ore is about 35 per cent., the fines being much higher.

The acid works have been successfully operated since their completion in July, 1907, and machinery has been installed at various times to increase the capacity and to make new acids. At present sulphuric, hydrochloric, nitric and mixed acids are made by the contact process and shipped in the company's tank cars to various parts of Ontario and Quebec.

Electric power supplied by the Seymour Power and Electric Co. is used throughout the mine and acid works.

The Canada Mine, No. 8, which was formerly the Oliver prospect, adjoins the mine operated by the Nichols Copper Co. on the east, and is located on the same fahlband. The lode strikes east and west and dips 50° to the south. During part of 1907, the Canadian Pyrites Co. sank an inclined shaft on the deposit to a depth of 110 ft. and did some drifting on the 85-ft. level, together with some diamond drilling. The deposit varies from 4 to 7 ft. in width. The ore on the dump is pyrite with a little pyrite and pyrrhotite, which will grade upward of 40 per cent. in sulphur.

The Hungerford Western Extension, No. 9, was fairly well prospected in 1906 by means of surface trenches at regular intervals along the strike of the fahlband. The western lens had been exploited by surface trenches to a length of 500 ft., exhibiting, near the line between the lots, a width varying from 16 to 18 ft. of ore, which will grade from 42 to 44 per cent. sulphur. The only impurity consists of small included lenses of calcite.

The eastern lenses are presumably continuations of the Hungerford mine orebodies.

A gossan 40 ft. wide occurs on the south end of the property, but not enough work has been done to determine the extent of the deposit.

The Ontario Sulphur Mines, Limited,[9] *No.* 10, commenced work in

[9] Formerly the Craig property.

March, 1908, and continued until the end of 1911, save for 2 months in the summer of 1910. The pyrite deposit on which work has been done is located about ½ mile west of the Hungerford mine. It appears to be a lens pitching toward the southeast. A shaft has been sunk 300 ft., with 225 ft. of drifting on the 100-ft. level and 250-ft. on the 200-ft. level. According to A. W. G. Wilson,[10] "The total shipments from the property up to the first of May, 1911, have been 4,821 long tons of ore averaging 36½ per cent. sulphur." In one place the deposit is 30 ft. wide.

The Sulphide Chemical Co. operated the property from the spring of 1914 until the following November, during which time the mine was dewatered and considerable ore was raised and shipped.[11] No work has been done since.

The Queensboro Mine (Blakely), No. 11, up to the autumn of 1906 shipped 65 carloads of pyrites running about 45 per cent. sulphur. Mine operations ceased in 1908. The pyrite occurs as a series of lenses up to 15 and 20 ft. wide along the contact of a garnetiferous schist (Grenville in age) and an intrusive pink felsite (post-Hastings in age). The ore is dense, the only impurity being thin veinlets of quartz. Cutting a pyrite lens is a small quartz vein containing copper pyrites and argentiferous jamesonite. In another place some zinc blende is interbanded with the pyrite. The main shaft is 135 ft. deep with about 175 ft. of drifting on the 50- and 85-ft. levels.

The Canadian Sulphur Ore Co.'s Pyrites Mine,[12] *No.* 12, was discovered in 1906 by Stephen Wellington while prospecting for iron. Under the gossan, merchantable iron pyrites was discovered, from which a car load of iron pyrites was shipped in 1908. Later, the Canadian Pyrites Syndicate bought the property, installed a small plant and shipped a few hundred tons of pyrite. In the spring of 1910 the property was handed over to the present company, which began shipping ore 3 months later, and has continued to the present. The mine is equipped to produce 100 tons of iron pyrites per day, yielding 40 per cent. of sulphur. Since Dec. 11, 1912, the mine has been run by electricity supplied by the Seymour Power Co. A branch line 2½ miles in length from the Bay of Quinte Railway near Queensboro to the mine was completed in 1913. The ore is shipped to the Nichols Chemical Co.'s acid plant at Sulphide, 11 miles southeast, and to the chemical companies at Hamilton and Detroit.

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[10] A. W. G. Wilson: Pyrites in Canada, *Publication No.* 167, *Canada Mines Branch,* p. 66 (1912).

[11] A. W. G. Wilson: *Annual Report, Ontario Bureau of Mines,* vol. 23, part I, p. 174 (1914).

[12] Alex. Longwell, President, 410 Crown Office Building, Toronto. For a fuller description see P. E. Hopkins: The Queensboro Iron Pyrites Deposits, *Annual Report, Ontario Bureau of Mines,* vol. 22, part II, pp. 89–104 (1913).

The pyrite is mined by underground and open-pit methods. The development work consists of three shafts and two open cuts, with some diamond-drill borings. Nos. 1 and 2 shafts, which are 75 and 100 ft. deep respectively, have been abandoned for some time. The work in late years has been confined to shaft No. 3 and the two open pits. The vertical shaft, No. 3, is 250 ft. deep with about 800 ft. of drifting on the 60-, 120-, and 200-ft. levels. The pyrite deposits are marked by gossan outcrops from 2 to 30 ft. in depth. Beneath are the pyrite deposits, which occur as lenses in contact with rusty schist to the south and white quartzite to the north (both Grenville in age) near an irregular post-Hastings intrusion of gray felsite. The strike of the deposits is slightly north of east, while the dip is almost vertical, inclining slightly to the south. Lenses vary in width up to 25 ft., but horses of country rock are frequently inclosed in the pyrites.

The ore is high grade, very little cobbing, if any, having to be done. Ores have been shipped running 40 to 48 per cent. sulphur.

The deposits are free from impurities such as arsenic, zinc, lead, copper and calcium. The pyrite burns satisfactorily, and is in good demand by sulphuric acid makers.

The Davis or Palmer Deposit, No. 13, is in the Grenville limestone. On the surface the pyrite is 2 ft. in width; 9 ft. down there is said to be a deposit 15 ft. wide. A few carloads of pyrites were shipped from a pit sunk on the property.

The Farrell Deposit, No. 14, lies in and conforms with the schist which strikes northwest. Test pits for a distance of 200 ft. show either gossan or pyrite. A shaft has been sunk to a depth of about 25 ft. A sample collected by E. L. Fraleck, representing an average of 75 per cent. of the dump (which consists of about 40 tons), yielded 40.64 per cent. of sulphur. The deposit maintains a uniform width of 5 ft., the only impurity being crystalline limestone.

The McKenty Prospect, No. 15, shipped hematite 40 years ago. A pit at one time 60 ft. deep has caved in. An examination of the cull dump reveals the fact that all large lumps of apparent hematite have, when broken, a core of pyrites. In E. L. Fraleck's opinion, this is one of many instances throughout eastern Ontario where hematite constitutes the gossan capping of a sulphide orebody.

The Little Salmon Lake Deposit, No. 16, occurs in a chlorite schist which strikes east and west, the main rock in the area being a white crystallized limestone, probably of Grenville age. A trench, 40 ft. long, uncovered pyrite 15 ft. in width. An average of 75 per cent. of the pyritiferous material yielded 38.83 per cent. of sulphur.

On the *Gunter Property, No.* 17, a shaft, 20 ft. deep, has been sunk on a deposit consisting of alternating bands of quartz and pyrite 5 ft.

wide. A sample representing two-thirds of the dump yielded 39.50 per cent. of sulphur.

Snooks Prospect, No. 17.—A fahlband strikes northeast through a coarse, impure crystalline limestone, and can be traced across the adjoining lot 6 to Desert Lake. On the road allowance, 7 ft. of massive pyrite and 25 ft. of pyrite mixed with crystalline limestone were uncovered in obtaining material for the road.

On the *Stalker Prospect, No.* 19, is a well-defined fahlband, containing some hematite, and striking east and west. A small test pit has been sunk on a lens of pyrite which shows at that point a width of 6 ft.

The Foley Deposit, No. 20, occurs in an outlier of crystalline limestone surrounded on all sides at short distances by granite. The irregular deposit consists of small masses of pyrite and pyrrhotite in about equal proportions. The work consists of a pit, 80 ft. long, 40 ft. wide and 10 to 15 ft. deep, sunk on pyrite and pyrrhotite in about equal proportions intermixed with pyroxene, calcite, mica and molybdenite.

The Caldwell Prospect, No. 21, was opened in the fall of 1915 by Thomas B. Caldwell of Lanark. About 500 tons of ore have been mined, but the sulphur contents are not known.

Diesel Engines Versus Steam Turbines for Mine Power Plants

Discussion of the paper of HERBERT HAAS, (p. 161).

HERBERT HAAS,—(communication to the Secretary*).—Fig. 1 plainly shows that the comparison of the steam-turbine and Diesel-engine plants was made on a basis of 6,000 kw. continuous operating load. The tabulation of costs for an 8,000-kw. load is merely done to show the relation of increased output on the unit power costs. Both plants would sacrifice their reserve capacity, which is to insure continuous operation, if operated to generate 8,000 kw. continuously; unless Mr. Hawkins advocates the operation of one of the turbines at 33.33 per cent. overload, retaining the other as a standby, which, besides overtaxing the turbine, results in a great loss of efficiency.

It would be well for Mr. Hawkins to qualify certain of his statements. Thus, under the caption "Diesel Engines, Selection of Units," he says that "two of the largest Diesel-engine builders take the position that, in the present state of the art, they would not attempt to put out units over 1,000 kw.;" he should qualify it by saying that "two of the largest *American Diesel-engine builders* take the position that, in the present state of *the art in the United States,* they would not attempt to put out units over 1,000 kw."

The reason that American manufacturers restrict themselves at present to units of that size is to be found not so much in a lack of experience in building large units as in a lack of a sufficient demand for them in the United States to justify the heavy capital outlay connected with the development of such engines. There are relatively few localities in the United States where fuel prices are as high as they are in the Southwest, and regions of high fuel prices are sparsely settled and have a very limited demand for power. Where *low* fuel prices obtain, steam turbines are active competitors of Diesel engines for *large* power plants, particularly when the station load factor is *low.* As most of the American manufacturers of Diesel engines work under license agreements with European manufacturers, which limit their market to the United States and its possessions and Canada, a further limitation is imposed upon them, leaving to European firms practically the entire world as a market.

The greater number of power plants range in size from 300 to 1,000 hp.; the average size of the central station, on a basis of all central stations in the United States, not greatly exceeding 500 kw.; in these sizes it is hard for steam engines to compete with Diesel engines, unless fuel oil prices are excessively high, and coal prices very low, or the exhaust steam is used for heating purposes.

* Received Mar. 19, 1917.

When Mr. Hawkins makes the statement: "While it is true that in Europe units in excess of 2,000 hp. have been built in the two-cycle type, they are not considered, even by the manufacturers, to be out of the experimental stage," he cannot be familiar with the development of large Diesel engines abroad, where a number of manufacturers build engines in units of 3,000 hp. and 4,000 hp. and engines of that size have been in successful operation for a number of years.

As is plainly stated in Fig. 1 and in Table 1, the units selected for the Diesel-engine power plant are four 3,000 hp. (2,000 kw.) Sulzer vertical single-acting two-stroke cycle engines. They are engines with six cylinders each and are built by Sulzer Bros. of Winterthur, Switzerland. Engines of this type are past the experimental stage and are in successful operation in different parts of the world.

Another item needing correction should be stated: the two 1,250-hp. Diesel engines installed at Tyrone, N. M., were built by Carels Bros. of Ghent, Belgium, and only the third unit was built by the Nordberg Mfg. Co., the licensee of Carels Bros.

Cost of Plant

As for Mr. Hawkins' "revised" figures of "Cost of Plant," I will say that, having been connected as engineer with one of the foremost European Diesel engine builders and also with the leading manufacturer of Diesel engines in the United States, I have cost data that Mr. Hawkins may possibly not possess. The costs as stated were conservative at the time the estimates were made, namely, late in 1915. They would not hold good at present as all materials have since then greatly risen in price.

Mr. Hawkins quotes my article in the *Engineering and Mining Journal* of Apr. 26, 1913, incorrectly. The plant there considered is of 1,200-kw. capacity, not of 1,400-kw. It comprises three 400-kw., not three 450-kw. units. These are single-acting, four-stroke cycle, four-cylinder engines and are not comparable in size or character of engine to a plant of four 3,000-hp. units of two-stroke cycle engines. Moreover, the higher cost of the small plant was partly due to special flywheel type of alternator, whereas the cyclic regularity of multicylinder two-stroke cycle engines is so good that standard type of generator can be coupled direct to the engine.

Fixed Charges

Mr. Hawkins' arguments, that the life of Diesel engines is an unknown factor, the present engines having been developed within comparatively recent years, might with equal force be made as regards large steam turbines using high steam pressures and highly superheated steam,

developed but recently. There are Diesel engines running today that have been in continuous operation for more than 15 years with practically undiminished economy. The amount of interest and amortization to be charged in any well-equipped power plant is greatly a matter of financial policy and not so much a question of the actual life of the plant. In our age of technical and industrial progress, plants lose their usefulness through obsolescence rather than actual deterioration, and the management with foresight favors high amortization charges, *i.e.*, short life, to provide a sinking fund for the replacement of obsolete with new efficient machinery.

With this principle in mind, the Diesel plant should be the one more favored of the two, as with the constant rise in fuel prices, the economic advantage is entirely with the Diesel engine plant for the conditions considered.

Fuel

Mr. Hawkins argues that a fuel consumption of 0.64 lb. per kilowatt-hour cannot be obtained with engines of the two-stroke cycle type of that size, and cites my article in the *Engineering and Mining Journal* of Apr. 26, 1913, in which I said that the fuel consumption of two-stroke cycle engines is 10 per cent. higher than that of four-stroke cycle engines. My figures in my later article do not in the least contradict statements made in the earlier one. All I can say is that well-built, four-stroke cycle Diesel engines, even of *moderate size,* will deliver one brake-horsepower-hour with a fuel consumption at or near full load not exceeding 0.40 lb., and two-stroke cycle engines with a fuel consumption of 0.44 lb., which is in accordance with statements made in the article referred to by Mr. Hawkins.

These are not fuel consumptions merely obtained during shop tests, but are secured in actual, continuous operation of Diesel engines. The following is the commercial performance of a Diesel-engine plant in the United States, secured with four-cylinder four-stroke cycle Diesel engines directly connected with alternating-current generators.

Fuel oil consumption per kilowatt-hour delivered to the switchboard:

At full	load, 0.590 to 0.610 lb., average 0.600 lb.
At three-fourths	load, 0.590 to 0.614 lb., average 0.602 lb.
At one-half	load, 0.686 to 0.703 lb., average 0.692 lb.
At one-fourth	load, 0.880 to 0.966 lb., average 0.932 lb.

The fuel consumption of 0.64 lb. per kilowatt-hour for the 3,000-hp. two-stroke cycle Diesel units is conservative, and is based on operating results with such units. This high economy, apart from being due to the large size of units, is brought about by a number of constructional features which greatly increase the mechanical efficiency of these engines.

Mr. Hawkins then goes on to say: "The above applies to full-load operation. At fractional load there will be a falling off in economy of the Diesel engine in the same way that there is for steam turbine."

This should be qualified by saying that there is a falling off in "the same way," *but not in the same amount,* as the overall efficiency of the steam turbine decreases at a greater rate at fractional loads than that of the Diesel engine.

One of the principal characteristics of the Diesel engine, to which it owes its attractiveness as a prime mover, is its "flat" fuel consumption curve at fractional loads. Thus between full and three-quarters load there is no appreciable difference in its fuel consumption. In modern high-grade engines this increase at three-fourths load rarely exceeds 1 to 2 per cent. and at half load is but 10 per cent. greater than at full load.

This phenomenal efficiency is brought about by an increase in the indicated thermal efficiency of the engine at fractional loads, which counteracts to a certain extent the loss in mechanical efficiency due to an increase in the internal work of the engine and the air compressor at fractional loads.

The economy of the steam turbine is only one link in the chain of the different factors affecting the overall steam-plant economy. The boiler-plant efficiency, the maintenance of a high vacuum, greatly dependent on an ample supply of cold circulating water for condensing purposes, and the efficient operation of the auxiliary and condenser equipment all affect the steam-plant economy. It is, therefore, a well-established fact that the continuous operating economy of a steam plant differs materially from records secured during performance tests, and that the percentage increase in fuel consumption at fractional loads is greater than that of Diesel engines. As the economy of the Diesel engine is two and a half to three times as good as that of the most economical steam plant at full load, a greater percentage increase in fuel consumption at fractional loads is, of course, more serious than the mere numerical figures indicate.

The conditions leading to the selection of the type of plant considered are fully enumerated on page 165 of my article and need not be repeated here. The engines will deliver 1 kw.-hr. with a fuel consumption of 0.64 at full and three-fourths load. At an overload of 10 per cent. there is also no material increase in fuel consumption, to change this figure. The Diesel-engine plant could therefore have swings of 1,500 kw. to 2,100 kw. from three-fourths load to 10 per cent. overload and the fuel consumption stated in my article would still hold good. Such engines can be started at a moment's notice, which is not feasible with steam turbines, unless they have previously been heated, which has to be done gradually, particularly where highly superheated steam is used.

Such load variations, however, occur but seldom in a plant supplying current to a mine, mill, and metallurgical works; the load is fairly uniform,

with a high load factor. If peaks resulting from hoisting operations are high, it will prove economical and increase the life of the power equipment, to absorb such peaks, either by the use of a flywheel generating set, by using compressed-air storage and air-operated hoists, or by following the plan outlined in paragraph 6, page 167 of my paper.

Gravity is no criterion of the availability of a fuel for Diesel-engine operation. A light fuel oil may be very undesirable, whereas a heavy fuel oil, even 12° Bé., may prove to be highly successful. The composition of an oil will be the deciding factor determining the usefulness of a fuel oil for Diesel engines, and gravity sheds little light on that.

Mr. Hawkins argues that "there is of necessity always a falling down in actual operation from the test results," and that the fuel consumption is therefore higher than that obtained on acceptance tests. The contrary is the case. After Diesel engines have operated for some time, their fuel consumption is lowered, due to an improvement of their mechanical efficiency with the greater smoothness of motion in all parts. Nor would "any slight wear of cylinder reduce the economy very materially due to the *enormous leakage* possible," as such a state of affairs would bring the engine automatically to a stop. "Enormous leakage" would fail to secure the high compression required to heat the air sufficiently for the ignition of the fuel, and would have to be corrected at once by the engine operatives.

Maintenance and Lubrication

From Table 1 it will be noted that two machinists with yearly wages of $3,360 form part of the regular operating force of the Diesel plant, to maintain engines at highest efficiency. This is directly chargeable to maintenance. Besides, it is usual for the engineers on shift to assist in any engine cleaning or repairs. As the 1 per cent. allowed for maintenance in Table 1 is based on the cost of the plant in America, this will take care of the higher price level here. Operating data in my possession, of plants in America and Europe, make the above allowance a fair average. It is obviously impossible to allow for all contingencies. An element of good or bad luck is a factor in the repair cost of any power plant. This element of risk is possibly best taken care of by insurance underwriters.

Steam Plant

The reasons for selecting 6,000-kw. units are plainly stated in paragraphs 1 to 3, page 165 of my paper. Paragraph 2, page 165, reads:

"The selection of two 6,000-kw. rather than three 3,000-kw. steam turbines is made on account of the greater efficiency of a 6,000-kw. over two 3,000-kw. units. *In plants with fluctuating* loads which can be supplied by one or more smaller units operating in parallel during different portions of the day, each unit operating at or

near its full-load capacity, a number of small units is justified; such diversified load conditions, however, apply mainly to central stations in cities, rather than mine, mill, and smelter power plants, which have invariably a high load factor during the entire day."

The above should suffice in answer to Mr. Hawkins' criticism regarding the selection of units.

Fig. 3, in which the factors influencing the selection of a Diesel-engine or steam-turbine power plant for a given plant cost, fuel price, and load factor are graphically shown, has no relation whatever to Fig. 1 and Tables 1 and 2.

The latter apply to small and medium-sized power plants up to 6,000-kw. capacity, are based on actual operating economy and not on performance tests. Overall efficiencies of 12 to 16 per cent. for steam plants of that size are the exception rather than the rule.

In conclusion, it must be said that neither maintenance nor lubrication of the modern high-grade Diesel plant are "excessively high," nor did I say that the lubrication expense may vary "from \$2 to \$4 per kilowatt-hour." I said that lubrication may cost from \$2 to \$4 per kilowatt-year (8,760 kw.-hr.). Better results than the lower figure are secured in high-grade engines that use forced feed lubrication, and filter and cool the oil before reusing it. While the lubricating-oil consumption used to be very high in Diesel-engine plants, improved construction and oiling systems have lowered that very materially, so that less than 1 gal. per 2,400 hp.-hr. suffices in good engines with proper attendance.

Maintenance may vary from \$1.25 to \$5 per kilowatt-year, depending on a variety of conditions, such as size of plant, load factor, make of engine, and very small plants may even have a higher maintenance charge. The deciding factors in the selection of either type of prime mover will be the plant cost, fuel price and load factor. Water conditions may also influence the decision.

Mr. Freyn's statement quoted by Mr. Hawkins reflected the status of the Diesel engine in the United States more than 5 yr. ago, when the building in America of the high-grade European Diesel engine had not even commenced. Since then conditions have very materially changed and a number of American manufacturers under license from European builders build high-grade engines. These are being introduced in increasing numbers at present, as their economy is well established and a matter of record for many years.

Stoping in the Calumet and Arizona mines, Bisbee, Ariz.

Discussion of the paper of PHILIP D. WILSON (p. 118).

CLARENCE M. HAIGHT, Franklin Furnace, N. J. (communication to the Secretary*).—In that part of Mr. Wilson's paper describing the Gilman cut-and-fill system, a few features do not appear to be fully explained.

In what way is the waste used for filling obtained? The text mentions that if waste is encountered when driving the filling raises, these raises are stopped until the fill is needed; that seems to suggest mining the capping for fill, which might make the backs of the stopes bad as the stopes approach the top of the ore. If, on the other hand, the fill is brought to the raises in cars, are the crosscuts that are necessary in ore or waste? Is the cost of this fill included in the stoping costs given?

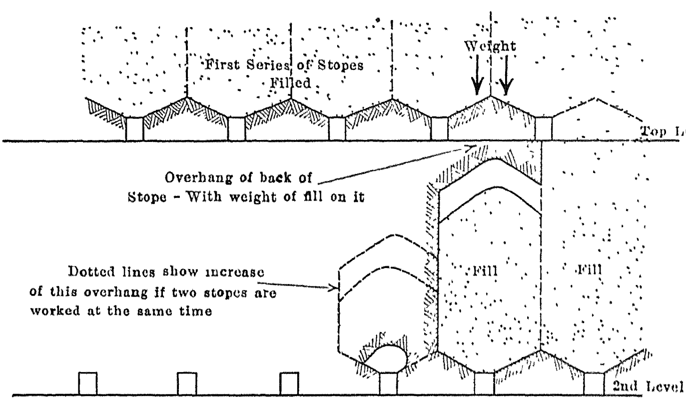

FIG. 1.

The second, or lower, series of stopes will present a different problem to that which the first series (under the capping) gave. Although no mention is made in the text, it would seem that the second series of stopes will have to be worked on center lines which are offset half the stope width, 20 ft., from the centers of the stopes above, in order to take advantage of the "rafter" effect of the stringers so named. How is the fill to be introduced into the lower stopes? There are no crosscuts mentioned to provide for this; or is it intended to tap the filled stopes above for the fill? In the latter case the method of control of this fill will be of interest.

Ore that will admit of a stoping width of 40 ft. with the lengths mentioned in the description is stronger than any with which many mining men are familiar. Has the practice of starting to mine a second stope before the preceding one is fully completed been tried to any extent? If

* Received Mar. 9, 1917.

so, has it worked as well as was anticipated? The stoping width in such practice becomes greatly increased in parts of the stope, and where a stope has already been finished, the top or back of the stope that is being worked becomes an overhang instead of an arch.

Mr. Wilson says that no stopes of the second series have as yet been brought up under the completed ones. Conditions under which the second series of stopes will operate when nearing the filled and completed ones above, will be greatly different from the original ones, especially after one stope has reached completion; the accompanying sketch will show this better than a description.

The dead weight of the fill in the old stopes will be much heavier than any encountered from an undisturbed capping. At Franklin, in the top slices, where the fill above the working places consists of rocks of all sizes packed only by their own weight, a heavy crushing effect due to weight is felt when only 20 ft. of length (about 3 sets wide) of a working place is exposed, and it is rare that the timbers will hold up for more than one set width more, about 27 ft. When this second stage has been reached and tried, doubtless many men who are actively interested in mining will be glad to learn how the system works out and how the ground behaves under the weight.

Is there a special reason for the use of the long stringers, *a*, rather than sets? As the chutes make it necessary for at least one post, *c* (and two are used when necessary) under each stringer, sets would require less timber and, being shorter, could be more easily handled. Two legs of 10 by 10-in. timber 8 ft. long and a cap of 12 by 12-in. timber 7 ft. long would give the same strength over the track and the same clearance. The legs would be on the solid, so no sills would be needed. All the other arrangements could remain the same; the 18-ft. rafter stringers could be braced against the legs and still reach to the side of the stope. Assuming both posts, *c*, as necessary under each stringer, the sets would require 108 ft. (b.m.) less of timber for every 5 ft. in the length of the stope; with only one post under each stringer, there would still be a slight saving in favor of the sets.[1]

If the cutting-out stope mentioned should be carried a few feet higher, the sets could be put in place and blocked temporarily, until the stope had reached its width and the rafter stringers placed. In this way the temporary timbering with "Cousin Jack" stringers could be eliminated. Unless the extra cost of mucking the additional ore broken to the floor by carrying the cutting-out stope a few feet higher exceeds the cost of erecting and dismantling the temporary timber work, the sets would appear to be worth trying, unless, of course, there is a special reason for the non-use of sets. Does it not happen in actual practice

[1] Corrections made on p. 133 of original paper would alter this paragraph. See author's discussion, p. 961, second paragraph.

that the cutting-out stope is carried higher than is actually necessary for the temporary timbering, so that in reality the "sets" plan would not mean any more mucking than at present?

PHILIP D. WILSON (communication to the Secretary*).—In replying to Mr. Haight's discussion, I will endeavor to take up the points to which he calls attention approximately in the order in which he mentions them.

The waste used for filling is obtained almost entirely from exploration or development work in barren ground. It is often possible to regulate the amount of such work by the need of filling in the stopes. The capping of a cut and fill stope is never mined for fill. Whether or not the crosscuts through which the fill is brought to the raises are in ore or waste depends entirely upon the shape and contour of the orebody on that level. The cost of handling and banking filling in the stope, usually insignificant except as the top of the orebody is approached, is included in the costs given. The cost of tramming and dumping is charged to development, legitimately enough as this cost is much less than the cost of tramming to the shaft, hoisting and dumping on the surface.

The second, or lower, series of stopes is worked on center lines directly beneath the centers of the stopes above. As yet no stopes of the lower series have been carried up to the sill of the stopes above. The intention is, however, to erect square sets on the consolidated fill, coming under the long horizontal stringers three sets or more wide. Timbering will be started when within 16 ft. of the level. When horizontal stringer is once caught up it will be a simple matter to come under the rafter stringers with stulls, or with square sets if necessary.

The main purpose of the long horizontal stringer is to provide a simple and safe means for catching up the filled stope above when the lower stope comes up beneath it. While this stage has not yet been reached in this type of stope, very similar conditions have been encountered many times in mining beneath large filled open stopes. It has been found that the filling, added in comparatively thin layers and subjected each time to the pressure of the broken ore, becomes by the time the stope is finished thoroughly consolidated and compact, in effect almost as strong as virgin ground, requiring blasting to loosen it. This would not, of course, be true of the thoroughly loosened matte above the Franklin top slices, instanced by Mr. Haight, where the dead weight of the fill is tremendous.

Filling is introduced into the lower series of stopes through raises which are holed into the original crosscuts on the level above. It is probable that the final 16-ft. square-setted section directly beneath the level will be filled by tapping the filled stopes above, but this has not yet been done.

* Received Apr. 5, 1917.

The practice of starting a second stope while the preceding one is still being mined has been quite successful. It has been found advisable to fill this second stope from a raise near the section line of the original stope with the peak of the cone consequently close to the gob lagging, and not to attempt to maintain an arch in the back of the stope.

The length of the horizontal stringers, *aa*, should have read 18 ft., that of the rafter stringers, *bb*, 14 ft. 2 in., in the original paper, page 133. Thus the apparent saving of timber by Mr. Haight's suggestion that sets be used instead of the horizontal stringer and posts is eliminated.

Furthermore, as stated above, the use of the latter makes it a far simpler matter to catch up the filled stope when coming up with a new stope from below. The use of the temporary "Cousin Jack" stringers has been a decided success. Their erection is very simple and rapid and any such means of substantially reducing the labor item is particularly valuable in a district where wages are so high. Cars can be loaded just as rapidly through the floor laid on the "Cousin Jack" stringers as from the permanent stope chutes.

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[NOTE.—In this Index the names of authors of papers are printed in small capitals, and the titles of papers in italics. Casual notices, giving but little information, are usually indicated by bracketed page numbers. The titles of papers presented, but not printed in this volume, are followed by bracketed page-numbers only.]

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